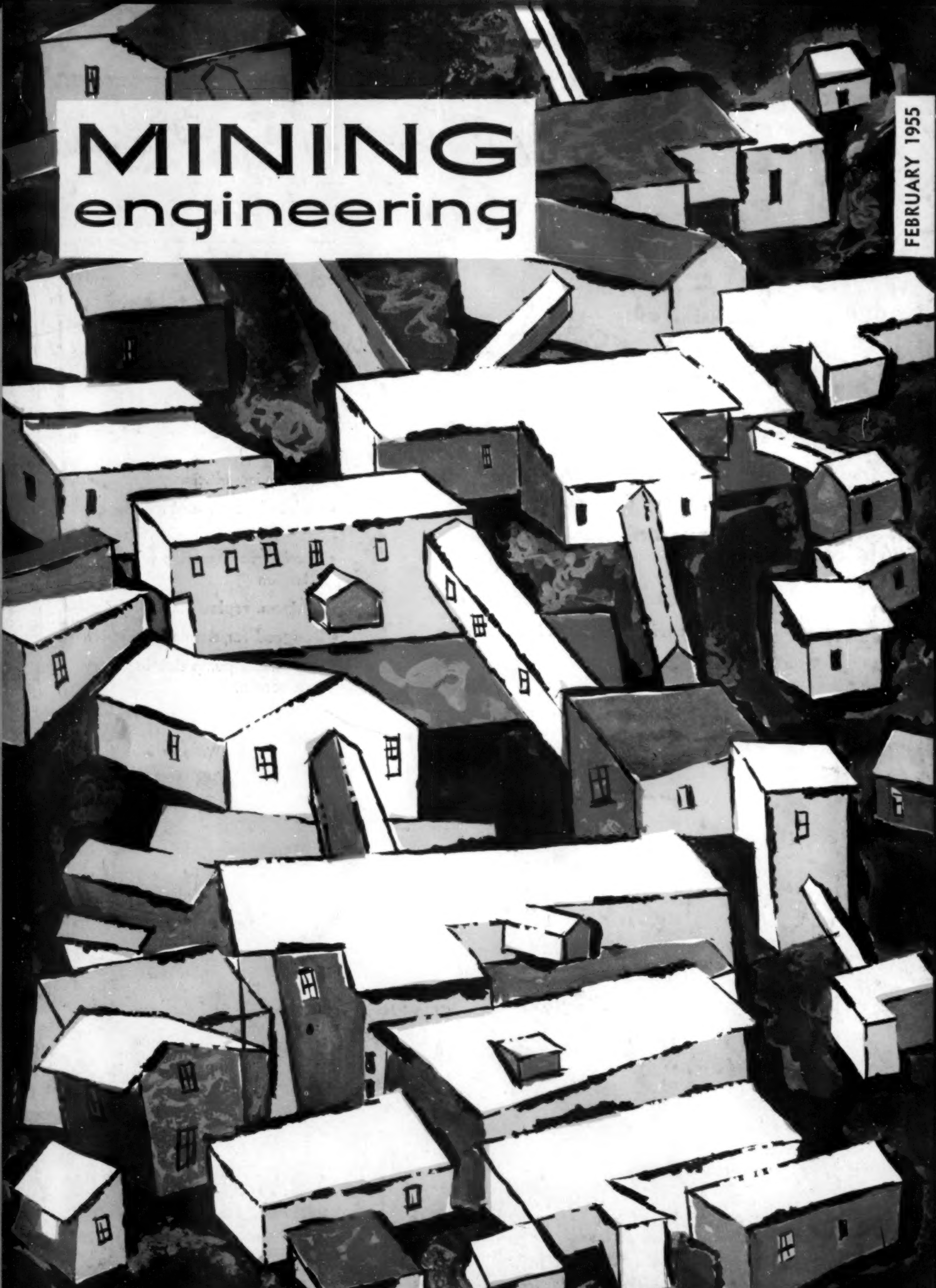
MINING engineering

FEBRUARY 1955



MINING engineering

VOL. 7 NO. 2 FEBRUARY 1955

The AIME publishes { Mining Engineering, Journal of Metals, Journal of Petroleum Technology

Published monthly by the American Institute of Mining and Metallurgical Engineers, Inc., 29 West 39th St., New York 18, N. Y. Telephone: PEnnsylvania 6-9220. Subscription $8 per year for non-AIME members in the U. S., & North, South & Central America; $10 foreign; $6 for AIME members, or $4 additional in combination with a subscription to "Journal of Metals" or "Journal of Petroleum Technology". Single copies $.75; single copies foreign $1.00, special issues $1.50. The AIME is not responsible for any statement made or opinion expressed in its publication. Registered cable address, AIME, New York. Indexed in Engineering Index, Industrial Arts Index, and by The National Research Bureau. Entered as second-class matter Jan. 18, 1949, at the post office at N. Y., N. Y., under the act of March 3, 1879. Additional entry established in Manchester, N. H. Member, ABC.

COVER

Impressions of the beneficiation plants in one well-known Mexican mining district provide the basis for artist Herb McClure's colorful cover. Problems now being faced by these Mexican producers, some with a history of over 400 years of metal output, are covered in the article starting on page 127.

FEATURES

ARTICLES

TRANSACTIONS

TECHNICAL NOTES

—Personnel Service—

THE following employment items are made available to AIME members on a non-profit basis by the Engineering Societies Personnel Service Inc., operating in cooperation with the Four Founder Societies. Local offices of the Personnel Service are at 8 W. 40th St., **New York 18;** 100 Farnsworth Ave., **Detroit;** 57 Post St., **San Francisco;** 84 E. Randolph St., **Chicago 1.** Applicants should address all mail to the proper key numbers in care of the New York office and include 6c in stamps for forwarding and returning application. The applicant agrees, if placed in a position by means of the Service, to pay the placement fee listed by the Service. AIME members may secure a weekly bulletin of positions available for $3.50 a quarter, $12 a year.

MEN AVAILABLE

Assistant Manager or Mine Superintendent, 46, married, will go single status. Twenty-five years underground metal mining, engineering and operating. Open pit and earth moving, shaft sinking, examination, exploration, and development experience. Registered engineer. Good understanding of labor relations. E.M. degree. Good Spanish. Available reasonable notice; prefers western U.S., Latin America, P.I. M-114-1122-E-San Francisco.

Mining Geologist, 38, married, M.Sc. Fifteen years varied experience exploration, development, operations. Good background in photogrammetry, geophysics, geochemistry. Has done extensive evaluation and report writing on nonferrous mineral deposits both U. S. and foreign. Available 30 days anywhere in U.S. M-115-1123-E-San Francisco.

Mining Engineer, 28, B.S., M.S. degrees in mining engineering, Reg. P.E. Married. Good scholastic record. Three and one half years experience chiefly in coal preparation with some iron ore beneficiation work. Desires permanent responsible position in ore beneficiation or related work, preferably East or Midwest. Currently employed; available reasonable notice. M-116.

Metallurgical or Mining Engineer: Graduate, 27, married, one child. Four years experience as engineer and supervisor in open-pit mining, crushing and milling operations of Fe. Desires position in engineering or supervision.
Box 1-E AIME
29 West 39th St., New York 18, N.Y.

POSITIONS OPEN

Mining Engineer, 35 to 50, with experience in both open cut and heavy equipment involved therein, and underground and shaft mining for production of industrial minerals. Salary, $7200 to $9000 a year. Location, North Carolina. W-873.

Mining Engineer, young, capable of carrying out exploration work and simple operation of open pit. Must have had 4 or 5 years experience in open pit work iron ore property—iron ore washing. Location, West Indies. W-845.

Mine Shift Boss, with good knowledge of Spanish, to supervise mining operation. Must go single status. Salary $4200 to start, $4500 after 3 months, plus room and board. Location, Central America. F-782.

Mining Geologist, Master's degree or Ph.D. Must have minimum of 10 years field experience. Must have good knowledge of theory and application of geophysics in order to evaluate findings of exploration teams who will report to him as manager. Salary, $10,000 to $15,000 a year depending on experience. Location, West. S-238.

Mine Superintendent, 30 to 40, for an underground mining operation, using modern Caterpillar propelled drill jumbos, diesel shovel loading equipment, and diesel truck haulage. The operation is producing about 2000 tpd. Salary, $6000 or more a year. Location, West. W-906.

Metallurgist, experienced in metallurgical processing; thoroughly familiar with milling equipment such as bill mills, jigs, classifiers, flotation, etc. Salary open. Location, Texas. W-870.

Industrial Engineer, under 40, preferably with experience in metallurgy, chemical or large milling operations. Salary open. Location, West. W-835(b).

Superintendent Manager, graduate mining engineer, with considerable experience in the economics of mining and milling, as well as experience in tunnel and open pit mining. Must speak Spanish. Salary, $8000 to $10,000 a year. Location, Cuba. F-900.

Meet The Authors

Wilbur T. Stuart (p. 148) has appeared in the pages of MINING ENGINEERING before, as the author of *Mine Drainage Studies in Michigan Iron Ranges.* He is currently hydraulic engineer in charge of mining hydrology research for the U. S. Geological Survey in Ishpeming, Mich. A member of the AIME, he received his early training at the University of Colorado, graduating with a B.S. in civil engineering. He has been working on industrial and irrigation water supply problems since 1931, and since 1945 on problems relating to water in mines of the Michigan iron ranges, Wisconsin and Nevada lead operations, Tennessee and Pennsylvania zinc facilities, Alabama iron mines, and Arkansas bauxite producers.

A. R. Kinkel, Jr. (p. 167) started his mining career as a sampler and miner in Butte, Mont. He has been a geologist and chief geologist with Hudson Bay Mining & Smelting Co., Flin Flon, Man.; chief geologist, Buffalo Ankerite Gold Mine, Porcupine, Ont.; and a geologist with the U. S. Geological Survey, a position he still holds. He is currently involved in a two-year study of the copper deposits of the Philippine Islands as part of a Foreign Operations Administration study of the country's strategic minerals. Mr. Kinkel received his A.B. from Stanford University. He has been an avid photographer for many years and also takes to the woods for some camping at every opportunity.

A. R. KINKEL, JR.

Robert H. Weber (p. 174), like a lot of men in the minerals industries, is a photographer, hunter, and angler in his free time. Most of his time, however, is taken up with his position as an economic geologist with the State Bureau of Mines and Mineral Resources, New Mexico Institute of Mining and Technology. He lives in Socorro, N. M. A graduate of Ohio State University and the University of Arizona, Mr. Weber earned his Ph.D. in 1950. He has been a geologist with Shell Oil Co. and photogrammetric and aerial photography officer with the USAAF. He is a member of Sigma Gamma Epsilon, geological fraternity.

Rollyn P. Jacobson (p. 158) is a research engineer with Sinclair Research Laboratories and lives in Tulsa, Okla. He earned B.S. and M.S. degrees at Washington University, St. Louis. In the past he has been associated with H. LeRoy Scharon as field geophysicist and geologist. Mr. Jacobson has had considerable experience in mining and engineering geophysics—particularly in electrical resistivity and magnetics. His hobby interests encompass photography and writing. Mr. Jacobson's article in this month's issue is his first published work.

GM DIESEL
CASE HISTORY NO. 5310-1X

OWNER: American Zinc Company, North Friends Station Mine, Mascot, Tenn.

INSTALLATION: GM 4-71 Diesels powering 3 Koehring Dumptors. Units haul 8¾ tons of rock and ore on 1100-foot run up 11½% grade to surface.

PERFORMANCE: GM Diesel-powered units help 20-man crew produce 500 tons of ore per day. Use of rubber-tired equipment cut cost of access shaft construction by 70%.

GM
GENERAL MOTORS
DIESEL POWER

GM DIESELS GO UNDERGROUND

to make trackless mining pay

One of the world's most efficient small underground mines, the American Zinc Company's new North Friends Station mine is proving the economy of trackless operation. With a fleet of three GM Diesel-powered Koehring Dumptors handling the haulage, the mine is producing 500 tons of ore per day—about 25 tons of ore per man-shift. Mine Foreman Bill Armstrong says: "These units run along month after month with almost no repairs. Our costs are much less than we expected."

General Motors 2-cycle Diesels are just as much at home underground as on the surface. Two-cycle operation with uniflow blower scavenging gives more complete combustion of low-cost fuel for higher efficiency and cleaner exhaust. Used with exhaust scrubbers and adequate ventilation, mine air stays well within permissable limits. And two-cycle operation means faster acceleration, quicker response to controls, faster haul cycles for increased production.

There's a GM Diesel for every kind of mining job. Specify GM Diesel power in *your* equipment. It will save you money.

DETROIT DIESEL ENGINE DIVISION
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Books for Engineers

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The Williston Basin, by Emory N. Kemler and W. D. Lacabanne, *Summary Reports*, $12.00, 272 pp., 1954.—A compilation of information widely scattered in petroleum magazines and professional society publications. It covers the geology of the Basin in general and of the various states and provinces; exploration methods in use in the Basin; drilling practices and problems; production, transportation, and refining activities; and characteristics of oil produced. Brief bibliographies follow some chapters and the appendix gives a general bibliography of 104 references. The authors are members of the engineering faculty, University of Minnesota.

Geology of Petroleum, by A. I. Levorsen, *W. H. Freeman & Co.*, $8.00, 703 pp., 327 illus., 47 tables, 1954.—Intended for students who have had basic courses in geology and also for petroleum geologists who are actively exploring for oil and gas pools. The material follows this order: the reservoir, with particular emphasis on the trap; reservoir conditions of temperature and pressure and different reservoir fluids; speculative ideas on origin, migration, and accumulation; and ways of applying what has been considered in the search for new pools and provinces. Practical applications are included along with theoretical analyses of the geological elements involved in petroleum exploration.

The Cupola and Its Operation, *American Foundrymen's Society*, Chicago, $9.50, 332 pp., 2nd ed., 1954.—A combined work by an impressive list of authorities that should be of value to gray iron foundries, which are operating more than 5000 cupolas in the U.S. and Canada. New material covers such late developments as hot blast, basic lining for nodular iron and emission control. Other chapters on refractories, principles of combustion and metallurgy have been greatly augmented. Bibliography, 328 illustrations, and 54 tables.

Materials of Construction, by M. O. Withey and G. W. Washa, *John Wiley & Sons Inc.*, $9.00, various pagings, 9¼x6 in., bound, 1954.—A revision of the authors' textbook published in 1939, which was based on Johnson's standard work of the same title. It covers sources, manufacture, and fabrication of materials; data on mechanical and physical properties; causes of defects and variations; testing methods; and general uses. A new chapter on concrete aggregates has been added and most of the other chapters have been rewritten or extensively revised.

Sonics, Techniques for the Use of Sound and Ultrasound in Engineering and Science, by Theodor F. Hueter and Richard H. Bolt, *John Wiley & Sons Inc.*, $10.00, 456 pp., 1955.—Growing research interest in determining the possibility of application of sonics to certain beneficiation problems makes this book of importance to mineral engineers. It is a thorough presentation of the principles and practices, covering the entire frequency range—from audible sound to ultrasonics.

The Geology of South Africa, by Alex L. du Toit, edited and prepared for publication by S. H. Houghton, *Hafner Publishing Co.*, $12.00, 611 pp., 3rd ed., 1953.—The present edition has been revised and enlarged to incorporate hitherto unpublished information assembled since the previous edition in 1939. Sections covering primitive formations, and in particular the Witwatersrand System, have been rewritten, and the accompanying geologic map and many of the figures in the test have been redrawn.

The Use of Stereographic Projection in Structural Geology, by F. C. Phillips, *Edward Arnold Ltd.*, London, available in the U. S. from St. Martin's Press, $3.00, 86 pp., 1954.—A simple, brief account, designed for the working geologist with no previous knowledge of the method. Principles are outlined and shown in application to the solution of various kinds of specific structural problems—determining true and apparent dip, interpreting cores from nonparallel bore holes, etc. The last chapter illustrates the usefulness of the method in synthetic studies of the tectonics of a region. There is an appendix on calculation by spherical trigonometry and a bibliography.

Nuclear Geology, A Symposium on Nuclear Phenomena in the Earth Science, edited by Henry Faul, *John Wiley & Sons Inc.*, $7.00, 414 pp., 1954.—Twenty-six authorities contributed to this book, which begins with a simple introduction to nuclear physics, followed by an outline of the more important techniques. The occurrence of radioactive elements in rocks and oceans is discussed and thermal, physical, and chemical effects of radioactivity are considered. One chapter outlines the nuclear methods of geophysical exploration and well logging, and another delves into some detail into techniques and results of absolute age determination. The last chapter discusses the origin of the earth.

Please Order the Books Listed Below from the Publishers

The Nation Looks at Its Resources, *Resources for the Future Inc.*, Mid-Century Conference Report Office, 1606 New Hampshire Ave., N.W., Washington 9, D.C., $5.00 (postage paid if payment is sent with order), xii + 418 pp., 1954.—A record of the conference held in Washington Dec. 2, 3, and 4, 1953 with 1600 participants. The "authors" are 536 men and women who spoke their views on: competing demands for use of land, utilization and development of land resources, water resource problems, nonfuel minerals, energy resource problems, world resources, resources research, and patterns of cooperation.

Industrial Hygiene Foundation of America Inc., Mellon Institute, 4400 Fifth Ave., Pittsburgh 13, Pa., has announced that reprints of the following papers are available free: *Exhaust for Hot Processes; Theoretically Required Exhaust Rates for Dust Control in Bulk Material Handling Systems;* and *Current Conceptions on Air Pollution* by W. C. L. Hemeon; and *The Magnitude of Errors in Stack Dust Sampling* and *A New Method for Stack Dust Sampling,* by W. C. L. Hemeon and George F. Haines, Jr.

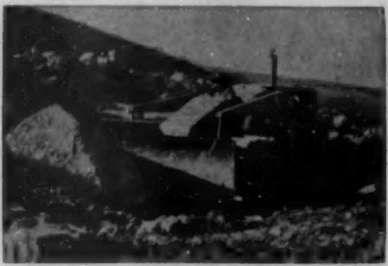
(Continued on page 108)

Welding for Engineers, by Harry Udin, Edward R. Funk, and John Wulff, *John Wiley & Sons Inc.*, $7.50, 430 pp., 230 illus., 1954.—Mr. Udin and Mr. Wulff are with the dept. of metallurgy, MIT, and Mr. Funk is an engineering consultant to Goodyear Aircraft Corp. Their book, a lucid presentation of the basic principles and practices of welding engineering, is devoted to the weld itself. It includes a thorough treatment of cold, hot pressure, and resistance welding, along with the permanent-electrode arc-welding and consumable-electrode processes. Welding with chemical heat sources is given careful attention, along with three full chapters devoted to brazing and braze welding. The concluding chapter describes principles of weld inspection and testing.

Some Fundamentals of Petroleum Geology, by G. D. Hobson, *Oxford University Press*, $2.90, 139 pp., 1954.—This volume is a survey of present knowledge of such basic problems as the nature of oil accumulation, composition and properties of reservoir fluids, the origin of petroleum, migration and accumulation, and reservoir pressure. Appendices describe compaction, in sediments, and define certain important terms.

Tractor-Dozer

Eimco Corp. now has its 105 tractor available for bulldozer, winch, or



other attachments including loading equipment. Design highlights of the 105 itself are: Unidrive transmission, independent track control, finger tip lever operation, high maneuverabil-

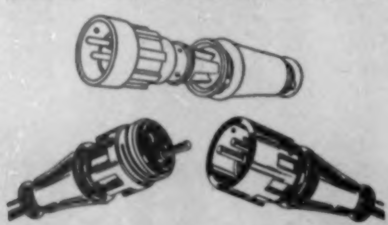
ity, and good operator vision. The 120-hp diesel-powered machines have matched torque converters and weigh 18 tons with dozer. Track widths to 24 in. are available for low ground pressure, and maximum drawbar pull is 45,000 lb at zero track slippage. **Circle No. 1.**

Pilot Plant Mill

Simplified pilot plant grinding is the story with *Denver Equipment Co.'s* Steel Head ball mill. Packaged mill and motor drive on single steel base, addition or removal of shell sections, plus variable discharge diameter provide complete flexibility to meet any problem. **Circle No. 2.**

Electrical Connectors

Joy Mfg. Co. has come up with the Quik-Loc line of electrical connectors



that can be mechanically attached to any cable. Other Joy connectors are of molded-to-cable design, vulcanized to short leads or cable lengths. New type attaches quickly, couples and uncouples in ¾ turn, and retains electrical features of other models. **Circle No. 3.**

Flexible Tubing

Neoprene coated nylon fabric is feature of Neolon flexible ventilation tubing recently introduced by *American Brattice Cloth Corp.* Puncture, acid, and oil resistant, and easy to handle, new tubing is available with two types of suspension, with various couplings, and in 8 to 36-in. diam. **Circle No. 4.**

Multi-Cyclone System

Higher efficiency is claim of the Paraclone mechanical dust collector developed by *Aerodyne Corp.* Feature is use of large number of small cyclones in parallel with secondary circuit to keep units under constant negative pressure. **Circle No. 5.**

G-E Locomotives

With two small locomotives now in production, *General Electric* has diesels and diesel-electrics ranging from 1½ to 250 tons in size. Shown are the 1½-ton, left, and the 10-ton diesel, center. The 1½-ton trammer on 18 or 24-in. gage track has normal 6-ft length that can be shortened to

4 ft for lowering on mine cages. The 10-ton unit is suited for switching and for use in open pit and quarry operations. **Circle No. 6.**

Nihard

Nordberg Mfg. Co., Milwaukee manufacturer of machinery for processing ores and industrial minerals, recently qualified as an authorized producer of Nihard, the nickel abrasion-resisting iron developed by International Nickel Co. Inc. **Circle No. 7.**

dc Motors

Super T dc motors introduced by *Reliance Electric & Eng. Co.* claim controlled reaction of motors to demand changes comparable only to previous specially designed motors. The 20 to 100-hp motors are said to have fastest and most accurate response ever offered. **Circle No. 8.**

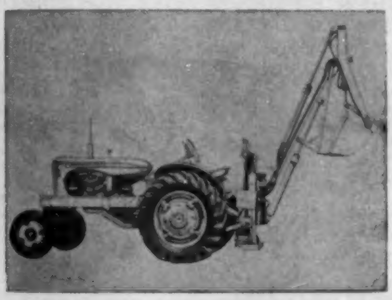
Refractory Gun

Designed specifically for air emplacement of granular basic refrac-

tories the *Basic Refractories Inc.* model A-20 gun offers facilities for fast economical repair of furnace linings. Operating platform and 20-cu ft hopper are new. **Circle No. 9.**

Wheel Tractor

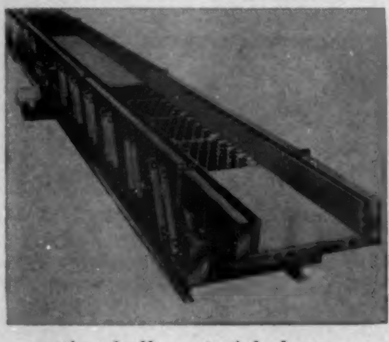
The tractor div. of *Allis-Chalmers Mfg. Co.* has added the WD-45 wheel



tractor with Henry backhoe to its line. Other attachments are available for the 45-hp, 4000-lb, WD-45. **Circle No. 10.**

Vibrating Conveyors

Syntron Co.'s line of balanced dual trough units for conveying and/or



screening bulk materials have compulsory drive, swinging mass design with high amplitude, and low frequency. Maximum capacity is 280 tph, and sections can be built up to 175-ft length. **Circle No. 11.**

Free Literature

(21) WEIGHING CONVEYOR: The Transportometer will make your ore conveyor a weighing conveyor. *Dwight-Lloyd's* bulletin 301 describes this accurate continuous weigher that is automatic and easily installed.

(22) CALCINE COOLER: Bulletin describes the *Stearns-Roger* improved calcine cooler that uses internal cells which approximately double the cooling surface—taking full advantage of the economy of water cooling both by conduction and evaporation. Features include continuous feed, shell protection, low power requirements, and predetermined cooling.

(23) STOPERS: Among features stressed in *Gardner-Denver's* leaflet on stopers is an exclusive air-operated water control valve and gland that gives automatic water with the "water on—air on, air off—water off" cycle required by certain local mining laws. This is valuable and time-saving, as well as a safeguard against dust concentrations.

(24) NEAR-SURFACE SEISMIC DATA: *Houston Technical Laboratories'* bulletin S-303 describes the HTL portable High Resolution seismic system that makes possible reliable reflection surveys over a depth range of 100 to 2500 ft. It is especially designed for use in mining, ground water location, petroleum exploration, and civil engineering where shallow seismic information is vital.

(25) SPRAY NOZZLES: For sheet flow spray arrangements in your flowsheet, *Deister Concentrator Co.'s* Concenco spray nozzles are easily installed without threading. Simply drill oversize holes, clamp on, and get results. Easily aligned, the nozzles are made in two sizes, for 1 to 2-in. pipe and for 2 to 4-in. pipe.

(26) GRIZZLIES: Information from *Nordberg Mfg. Co.* covers Symons vibrating bar grizzlies for heavy duty large capacity primary scalping from 1½ in. upwards and Symons vibrating rod grizzlies for big capacity scalping from ⅛ up to about 4 in. They have gained "an enviable reputation for dependability, efficiency, and economy."

(27) CONICAL SCRUBBER: Rub a handful of pebbles between your palms under a stream of water. This is similar to "Mass Action" in a *Hardinge* conical scrubber. Bulletin

37-A-2 describes this scrubber, which is capable of removing a very hard clay form ores, phosphate rock, gravel, etc., rapidly—but with only nominal wear in the machine itself.

(28) CRUSHERS: With the Speed-Set control on the Hydrocone, dismantling and auxiliary equipment are not required to change from one product size to another. *Allis-Chalmers'* bulletin 07B7145B shows how this hydraulic adjustment provides "one man . . . one minute product control."

(29) JAW CRUSHERS: Bulletin 125 from *Traylor Eng. & Mfg. Co.* outlines the exclusive design features of Type S jaw crushers, the most massive jaw crushers ever built. Available in seven sizes, with feed openings from 36x42 to 60x84 in., crushers offer capacities up to 1000 tph.

(30) H-M PUMPS: Complete details are available from *A. R. Wilfley & Sons Inc.* on pumps that deliver continuous, trouble-free performance at low cost without media loss, leakage or dilution. Features include individual engineering on every application, minimum replacement, and simple installation.

(31) DROP BALL: Price and application information data is now available on *Cape Ann Anchor & Forge Co.'s* forged steel drop ball for economical secondary breakage. Drop ball is made in sizes from 2000 to 12,000 lb, and longer life and better wearing qualities are due to forging from steel, stress relieving, and heat treatment.

(32) POTASH STORAGE: Field report No. 227 from *Sauerman Bros. Inc.*, engineers and manufacturers, describes the handling of potash in indoor storage at Carlsbad, N. M. It is well illustrated with storage layout drawings and pictures showing the plants of two major potash producers.

(33) GOLDFISH TO CONCRETE: The first basic patent in many years on any centrifugal type pump has been issued to the Wemco Torque-Flow solids pump. It handles anything from live fish to ready-mixed concrete. *Western Machinery Co.'s* bulletin No. SP-10 shows how the difference is in the recessed impeller that insures unobstructed passage.

(34) NICKEL ALLOY CAST IRONS: *International Nickel Co.'s* "Heat Treatment Fundamentals of Nickel Alloy Cast Irons" shows how to increase the usefulness of cast irons and improve the properties by alloying and heat treatment. A glossary of terms is included for easy understanding.

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61	62	63	Students should write direct to manufacturer.						

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(35) MAINTENANCE: Want to get the best service out of insulation, packings, refractory products, roofings, and friction materials? *Johns-Manville's* "Good Operating Practices" contains 101 suggestions for better maintenance.

(36) PACKING SELECTOR: *New York Belting & Packing Co.'s* 6-in. cardboard disk gives packing styles, pressure ratings, and temperature readings for four applications: gaskets, valve steam, centrifugal and reciprocating pumps. Simply rotate a smaller disk and correct packing is shown for material being handled: steam, acid, petroleum, brine, etc.

(37) HAULING PROBLEMS? Bulletin TK-170 describes *LeTourneau-Westinghouse* rear dump Tournapull designed specifically for the mining industry. Mechanical details include big low pressure tires, simple construction, instant dump, all-steel body, 90° turn, and provision for quick, safe stops.

(38) PROCESS INSTRUMENTATION: *Fischer & Porter Co.'s* 12-page general catalog covers a complete line of process instrumentation: flow meters, pressure instruments, electric, pneumatic, and electronic transmitting, recording, and controlling instruments.

(39) ALL-ELECTRIC SHOVEL: Bulletin from *Harnischfeger Corp.* introduces the P&H Model 1055-E, similar in size to the standard 3½-yd P&H Model 1055, but completely electric, driven by constant speed ac motors throughout. Power transfer is through frictionless P&H Magnetorque units for all shovel motions—swing, hoist, crowd, propel.

(40) FROTH FLOTATION: "Mineral Dressing Notes," bulletin No. 21 from *American Cyanamid Co.'s* mineral dressing dept. is an authoritative paper, "Froth Flotation," by Robert B. Booth. It was originally published in 1953 as a chapter in "Foams: Theory and Industrial Application."

(41) DRAWPOINT LOADING: The L-1017 series of publication from the *Eimco Corp.* contains numerous articles from mining journals and other material on drawpoint loading. This economical production method has numerous advantages in lower costs, higher production, and greater safety.

(42) GUAR BEAN FLOCCULENT: There's a new *General Mills* product that undoubtedly won't turn up at your corner grocery. It's Guartec, an ore dressing flocculent or depressant

made from Guar beans, and the story of new chemical is told in the company's quarterly, "Progress Thru Research."

(43) DIESEL IMPROVEMENTS: "The Inside Story" from Detroit Diesel Engine Div., *General Motors Corp.*, describes new cylinder liner and a new piston that results in a completely new cylinder assembly for series 71 engines.

(44) BELT CLASSIFIER: Among the features stressed in *Denver Eqpt. Co.'s* bulletin on the Denver-Finney submerged belt classifier are quiet pool area at any mesh separation, ability to stop-and-go under full load, and reduced overall maintenance.

(45) ENGINES: *Buda Co.'s* 21-page bulletin 1409 announces a choice of 190 basic engine models—industrial, automotive, marine, and electric generator sets—from the "smallest to the biggest."

(46) SUSPENDED MATERIAL RECOVERY: Operating principles and typical installation flowsheets of the Colloidair process are given in catalog from *Bulkley, Dunton Processes Inc.* Applications shown include mining and ore processing.

(47) MINING & QUARRYING STEELS: *Bethlehem Steel's* 20-page booklet 345 is divided into five sections: hollow drill steel, ultra-alloy hollow drill steel, solid and auger drill steel, broaching and channeller steel, and stone-dressing steel. Listed are typical analyses, instructions for working, heat treatment, characteristics, and uses.

(48) SIDE DUMP CARS: Bulletin SD-1 from *C. S. Card Iron Works Co.* covers Granby type side dump cars. Popular designs include a 30-in. gage eight-wheel car with 215-cu ft capacity, a car for all-year operation at high altitude, and an air operated car that uses no dump block or roller. Twin air cylinders cradled in the frame synchronize with an automatic rail hook that keeps the car upright while dumping.

(49) STAND-BY POWER: *Le Roi Div.*, Westinghouse Air Brake Co., has an 8-page bulletin describing "custom built" engine-generators for dependable low cost stand-by power. Illustrated and described are seven engine models from 60 to 750 hp with 50 to 350-kw generators.

(50) MAGNETIC FLOW METER: Bulletin presents specifications, photographs, and other data on the *Foxboro* magnetic flow meter. This meter measures the volume flow rate of many liquids and semiliquids "with freedom from many of the limitations of conventional flow meters of the differential pressure or variable area type."

(51) SECTIONAL BELT CONVEYOR: *Stephens-Adamson Mfg. Co.* now offers a complete set of belt conveyor components pre-engineered for customer assembly. Bulletin 1454 describes these Saco sectional units that are available in both 18 and 24-in. belt widths. Saco conveyor components are available from stock as package units for complete conveyor installation or as replacement units on existing conveyors.

(52) SCRAPERS: Available from *Sauerman Bros. Inc.* are two field reports. "How to Dig a Clean Pit" describes an arrangement at a Canadian mine that allows a 4-cu yd Sauerman scraper to supply sand and gravel directly to the shaft. "Handling Ilmenite Ore with Drag Scraper" shows the operation of a ¾-cu yd Sauerman scraper equipped with monorail and trolley for rapid shifting about a 146x311-ft storage area.

(53) TIMBERING COSTS: "Force Down Your Operating Costs" is 16-page booklet that shows and tells about the *Osomosalts* (timber preserving) treatment. See page 102.

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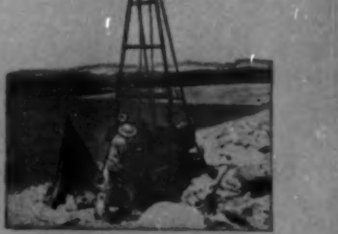

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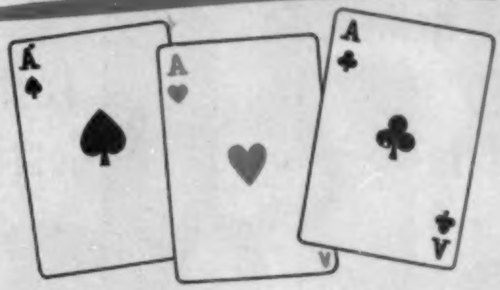

22-HD The rugged, heavy-duty model. Capacity—2000' with EX fittings. Also available as truck-mounted drill or on twin-column mount for underground operation. Bulletin D-28.

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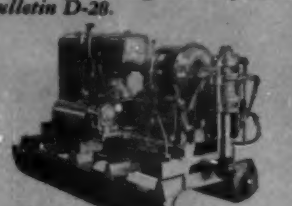

12-B Extremely portable, weighs approximately 1200 lbs. Capacity—1000' with EX fittings. Also available on twin-column mount with air motor drive for underground operation. Bulletin D-21.

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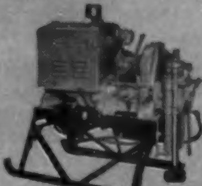

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(Continued from page 103)

Geology of Coshocton County, by Raymond E. Lamborn, *Div. of Geological Survey*, Room 106, Orton Hall, Ohio State University, Columbus 10, Ohio, Bulletin 53, $3.00, 245 pp., maps, 1954.—A physical description of the sequence of rocks in Coshocton County which include sandstone, limestone, clay, and coal, with an explanation of their economic utilization. John H. Melvin, chief of the Div. of Geological Survey, points out that this publication is of particular interest now, because of the current activity in exploration for oil and gas in the western part of the county.

Legal Guide for California Prospectors and Miners, *California Div. of Mines*, Ferry Bldg., San Francisco 11, Calif., 25¢, 78 pp., 3 fig., 1954.—Originally issued as a supplement to the July 1952 issue of *Mineral Information Service.*

Fifth Empire Mining and Metallurgical Congress Australia—April 1953. These volumes deal in an analytical and comparative manner with the mineral deposits and mining and metallurgical practices in Australia, New Zealand, and surrounding territories. They are available from the *Fifth Mining and Metallurgical Congress*, 399 Little Collins St., Melbourne C1, Australia. Prices quoted in next column are approximate.

The Geology of Australian Ore Deposits, Vol. I, edited by A. B. Edwards, $10.16 cloth cover, postpaid, 1100 pp., 1953.—A comprehensive description of the important productive orebodies, past and present. Structural control features are emphasized.

Mining Methods in Australia and Adjacent Territories, Vol. II, edited by R. Pitman Hooper and A. B. Black, $4.80 paper cover, postpaid, 350 pp., 1953.—An analysis of the application and adaptation of several accepted methods of mining and development for the extraction of orebodies, alluvial deposits, etc of Australia, New Guinea, and Fiji.

Ore Dressing Methods in Australia and Adjacent Territories, Vol. III, edited by H. H. Dunkin, $3.27 paper cover, postpaid, 350 pp., 1953.—Ore dressing processes as practiced in Australia, New Guinea, and Fiji are described with comments on the factors affecting the choice of process. Sections relate to lead, zinc, copper, gold, tin, tungsten, the important beach sand deposits of zircon and rutile, and several other minerals.

Extractive Metallurgy in Australia, Ferrous Machinery, Vol. IV A, edited by J. C. Richards, $4.23 paper cover, postpaid, 217 pp., 1953. **Nonferrous Metallurgy Vol. IV B,** edited by Frank A. Green, $3.03, paper cover, postpaid, 272 pp., 1953.—These volumes deal with smelting, hydrometallurgical, and other processes used in the production of iron, steel, ferroalloys, lead, zinc, copper, and other metals; and refer also to the utilization of the sulphur content of Australian ores for production of sulphuric acid and fertilizers.

Australian Mining and Metallurgy, Miscellaneous Features and Practices, Vol. V, edited by A. W. Norrie, $3.03 paper cover, postpaid, 300 pp., 1953.—A collection of articles on ventilation, safety, hygiene, mining regulations, industrial relations, education, training, and research.

Coal in Australia, Vol. VI, A Symposium Arranged by a Committee of The Australian Institute of Mining and Metallurgy, $7.18 paper cover, postpaid, 832 pp., 1953.—This covers in detail the important black coal resources of New South Wales and Queensland, the remarkable brown coals of Victoria, and the lesser coal deposits of South Australia, Western Australia, and Tasmania. Given are geology and seam constitution, mining and preparation practices, marketing and utilization, etc.

Handbook, Australia and New Zealand, edited by M. Glen, $5.95, cloth cover, postpaid, 250 pp., 1953.—A guide to the geography, geology, history, economy, and general social environment of Australia and New Zealand, with particular reference to mining and metallurgical industries. It also contains a record of the main features of the Congress.

CONSTRUCTION AND MINING equipment used during the building of the Kitimat-Kemano aluminum project in British Columbia is going on sale. The S&S Machinery Co., 140 53rd St., Brooklyn, will manage the sale of equipment, said to be worth some $24 million and consisting of more than 50,000 items. Shovels, tractors, cranes, dump trucks, drills, locomotives, flatcars, and even a fleet of helicopters are included. Unusual aspect of the sales program is a rental plan that makes most of the heavy equipment available under a leasing arrangement.

Overall value of Canadian mineral production in 1954 jumped to an all-time high of $1.454 billion in 1954, impelled by new peak values for the four main groups, metals, nonmetals, fuels, and structural materials. The total was $118 million more than the $1.336 billion in 1953. Metal production was valued at $763 million last year.

U. S. TREASURY DEPT. announced discovery of some sales of East German potash made at less than "fair value." The Tariff Commission has the case now and is attempting to determine if these sales are injuring or threatening the domestic industry. The Treasury also announced investigations into possible dumping involving other European countries. If the Tariff Commission finds that sales of imports at less than fair value are hurting the domestic industry, it can impose duties. Potash has never been protected by a duty. Complaints leading to Treasury action stemmed largely from five New Mexican firms.

Rutile has jumped into third place in importance in New South Wales, Australia, ranked by only coal and gold. New South Wales and Queensland are Australia's major producers of the yearly rutile export of 39,000 tons.

THE McDOWELL Co. of Cleveland has been awarded a contract to construct an agglomerating plant for Cleveland Cliffs Iron Co.'s Republic mine. The 2000-tpd plant will pelletize high grade iron ore concentrates from the jasper formation by the so-called updraft traveling grate process. The plant will be located on the main line of the Lake Superior & Ishpeming RR at a site a few miles west of the port of Marquette. The Republic mine is Cleveland Cliff's second venture in the development and concentration of Michigan's iron bearing low grade hematite. It is scheduled to go into production late this year.

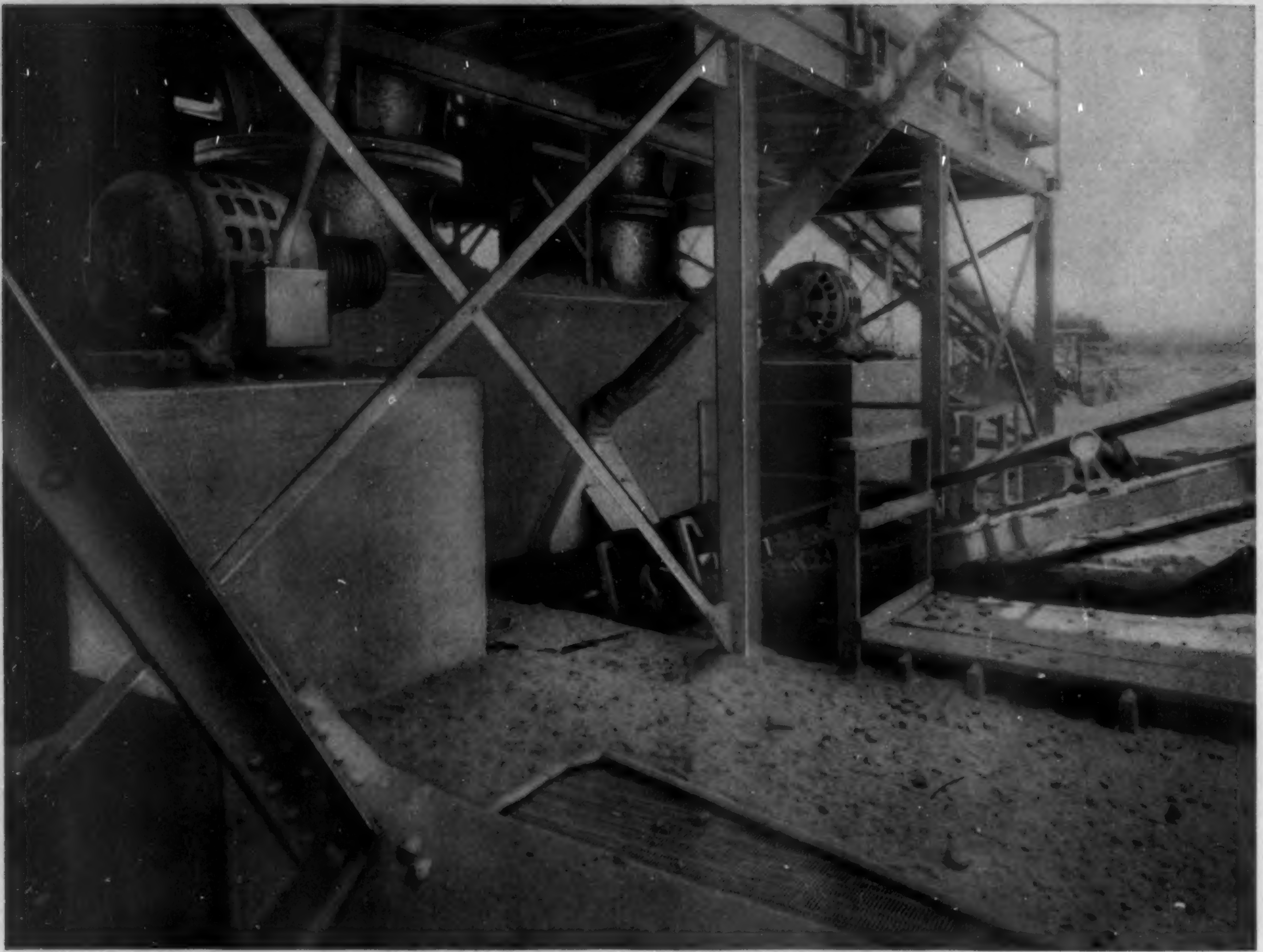
Lithium Corp. of America formally opened its new $7 million plant at Bessemer City, N. C. The plant will treat ore from the company's open pit at Kings Mountain and will also process lithium concentrates from other sources. The lithium compounds from the new facility will go to the general market.

SOME OF THE NATION'S uranium producers got together in Washington, D. C., and formed the Uranium Assn. of America, with Jack Turner, one of the more fabulous rags to riches personalities, elected president. Membership is expected from all segments of the uranium industry, including prospectors, miners, developers, producers, mill operators, investors, underwriters, and manufacturers. Membership of the former Inter-Mountain Uranium Assn., a Utah group, is joining the association.

New Jersey Zinc Co. and the Texas Co. will undertake exploration activities on the Colorado Plateau and other mineralized areas of the U. S. Exploration will be done jointly for the present. Should a uranium find be made with sufficient deposits to justify mining, a separate company may be formed.

VITRO CORP. OF AMERICA together with Rochester & Pittsburgh Coal Co. have formed a jointly owned subsidiary, Vitro Minerals Corp., for exploration, drilling, and mining of uranium claims held by the two parent firms. Interests include claims in the Gas Hills of Wyoming, holdings near the San Rafael Swell district of east central Utah, and shares in claims in the Blind River district of northern Ontario.

ENROLLMENT IN THE National Safety Council Mining Section's Prevention of Falls of Ground Accidents campaign is rolling along, with enrollment heavy. A great many Canadian mines, especially in Ontario, have joined up. In the U. S. many of the larger companies have enlisted and small operations in Oregon, the Southwest, Middle West, and East are also participating. The drive is being backed by various government and private agencies, the AIME among them.



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Recent tests showing approximately 3% increase in both copper and silver recovery at a Mexican flotation plant prompted adoption of AEROFROTH 80 Frother in place of cresylic acid. At this property silver occurs in pyrite as well as with copper-bearing sulfides. A bulk float recovers both minerals.

During a recent six-day plant test, cresylic acid was used on one bank of cells, AEROFROTH 80 on the other bank. Both were fed at the rate of 0.15 lb. per ton of ore. Other reagents used are 0.5 lb. soda ash, 0.28 lb. AEROFLOAT® 242 Promoter and 0.2 lb. potassium ethyl xanthate per ton of ore. Results were:

	Cresylic Acid				AEROFROTH 80 Frother			
	Assay		% Recovery		Assay		% Recovery	
	% Cu	Oz/Ton Ag	Cu	Ag	% Cu	Oz/Ton Ag	Cu	Ag
Head	4.29	4.41			4.88	4.74		
Concentrate	28.78	28.53	93.98	90.64	28.55	27.19	96.23	93.38
Tailings	0.30	0.50	6.02	9.36	0.22	0.38	3.77	6.62

The Cyanamid Field Engineer who worked with this operator reports:

"Higher recoveries with AEROFROTH 80 were due to more rapid flotation. The bank of cells fed AEROFROTH 80 carried practically no mineral to the last cell, flotation being complete in the preceding cells. With cresylic acid, sulfide flotation was considerably slower. The froth in the last cell carried considerable mineral".

A Cyanamid Field Engineer will be glad to work with you to determine whether AEROFROTH 80 Frother or other Cyanamid Reagents can increase recovery or cut costs on your mill. A note to our nearest office will have prompt attention.

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Republic Rutile Discovery May Depress Price

Possibly, the most significant aspect of Republic Steel Corp.'s new rutile operation in the back country of southwest Mexico is that the gigantic discovery may eventually force the price of titanium down as much as $1 per lb.

At least 25 million tons of the titanium bearing ore has been discovered in a region easily one of the most forbidding in all Mexico. Transportation into the district, 2 miles from the village of Pluma Hildago in the state of Oaxaca, is strictly by jeep or light truck. An automobile wouldn't stand a chance on the narrow, twisting roads. During the rainy season big pieces of the so-called roads are washed into arroyos thousands of feet below.

The discovery was made by Donald B. Gillies, retired Republic vice president, and now mining consultant to the corporation, and his aides, Ward H. Broadfield and Willis A. Seaman. The titanium dioxide content of the rutile is reported to be between 95 and 97 pct. The first mine to go into operation, Las Minas de Tisur, has proven reserves of 25 million tons. But it represents only a small part of the 38 claims Republic holds in an area 6 miles long and 1½ miles wide. Some 2000 tons per month will be shipped in drums and boxes, although some possibility exists that elevated storage bins will be constructed at Puerto Angel, 12 miles from the mine, for direct loading into ship holds.

The Tisur vein is along the face of a 4500-ft mountain. Hand labor shaved off the top of the mountain for a camp site and the first exploratory tunnel, adit No. 1 was driven into the vein 300 ft below the camp. At 300 ft the tunnel was still in ore. Crosscuts indicated vein width of at least 75 ft. Two more tunnels, at 100 and 200 ft lower elevation, were driven. Right now, 35 miners working two shifts are driving another adit where diamond core drilling offered evidence that the vein was more than 230 ft thick. A vertical hole 91 ft deep stayed in ore. Adit No. 4 will probably be used as the main entry, with ore from the other adits dropped down a vertical shaft to No. 4.

Ore will go by conveyor belt a half mile down the mountain to a concentrating plant where it will be crushed 60 to 80 mesh, washed to bring the TiO_2 content up to 95 pct. Diesel-driven generators will be used for power because a newly completed products pipeline at Salina Cruz 112 miles down the coast will make cheap fuel available.

The road to Puerto Angel is to be improved by the Mexican Government to handle 7-ton trucks but Republic will have to construct a 6½-mile connecting artery. Docks at the port will be rebuilt and extended by the Government.

Republic has no plans for building a titanium sponge plant at the moment, although it has complete setups for melting sponge and converting it into pure metal at Canton and Massillon, Ohio. Du Pont, Crament Inc., and Electromet seem likely customers for the rutile concentrate.

The possible drop in price arises from the partial release from dependency upon Australia and other countries for rutile.

Approve New Tractor For Mine Operation

U. S. Bureau of Mines granted approval for the first diesel track-type tractor especially designed for operation in non-coal underground mines.

Combustion characteristics of the Caterpillar Diesel D4 permit it to use excess air in the combustion of fuel and so that engine lends itself to safe operation. The "Cat" diesel engine meets the requirements at rated engine rpm that the engine's exhaust must be diluted to a low percentage of the ventilating air.

A specially designed exhaust conditioner, built by the National Mine Service Co., has been installed on the D4 tractor. This conditioner employs the water cooling principle to reduce exhaust temperatures to below 160°F, as required by the USBM.

Another feature of the unit is that the radiator fan has been reversed, blowing gases away from the tractor and immediately diluting them when discharged in front of the radiator through the diffuser. After what has been described as exhaustive testing, the tractor was certified for operation in non-coal mines.

The machine has been scheduled to go to work in an iron mine in the Birmingham, Ala., region. It is the first such unit to be given certification for safe operation under Schedule 24.

This is the Caterpillar D4 tractor recently approved by the U. S. Bureau of Mines for underground operation in non-coal mines. Shown looking over the features of the tractor are left to right (top), Fritz Jones, sales manager of the J. D. Pittman Tractor Co., Birmingham, Ala.; John C. Holtz chief, Diesel Test & Research Section, USBM; (on ground) Roger Davis, USBM chemist; and Robert James, chief, Development Section, USBM.

Rich Mica Strike In South Carolina

One of the biggest strikes in the recent history of mica-mining has been made in the southwestern corner of North Carolina near Cowee Township in agricultural Macon County which borders on Georgia and South Carolina.

The mining operations are under the direction of the Minerals Processing Co. of LaGrange, Ga., operators of the property.

Frank Siersma, president of Minerals Processing, stated that outside experts estimated the property to contain a large tonnage of good quality and valuable block mica.

Block mica, currently on the strategic minerals list, is being stockpiled by the General Services Administration. Sheet mica is also in wide demand by industrial users.

Block mica, also known as sheet mica when processed, is removed from the ground in book-like blocks. In the sheeting process, the sheets are separated according to quality and size. The degree of stain or discoloration determines the quality and worth.

The company has been developing the property for several months. "We knew it would yield mica, but the quantity and quality far exceeded the original estimates of our geologists," Mr. Siersma continued. "When we first started digging the Rose Creek shaft, we had planned a sheeting operation to handle an expected output of about 500 lb of block mica per day. As the shaft was sunk deeper, the mica started hardening up and the blocks became progressively larger. We're now set to handle 1500 lb per day, and are rapidly increasing our facilities to sheet 2500 lb of block mica per day."

Owners of the mica strike near Cowee Township in North Carolina report that the block mica yielded is of exceptionally high quality. Deposit is said to be one of the largest discovered in recent years.

The quality of mica removed to date has been exceedingly high. The first sheeted quantity prepared to Government stockpile specifications brought an average of $24.83 per lb compared to the average $13.00 per lb price paid by the Government's Spruce Pine, N. C., Purchasing Depot. A small portion of outstanding quality brought $70.00 per lb.

Besides its value to the Government stockpile, this strike will produce a large amount of mica in sheet, punch, and scrap form, available for a wide variety of industrial users.

Sheet mica was first termed isinglass and used for the windows in cooking stoves because of its ability to withstand heat. In later years, it became widely used for another type of window, side and rear sections in many touring cars. Today, it fills a vast need as an insulator in electric and electronic devices and is practically alone in satisfying this need. Wet ground and dry ground mica are used in paints, plastics, floor waxes, wallpaper, and rubber. Finely ground mica is combined with oil and grease for use as a lubricant.

Construction of Lithium Corp. of America's Bessemer City, N. C. plant is completed. The plant will process run-of-mine ore directly through the chemical plant, eliminating the concentration stage. Spodumene ore comes from company's Kings Mountain mine.

Rock Dumps Producing Millions

Mountains of rock dumps promise the revival of one of the most famous boom towns in Ontario—Cobalt—putting what was once one of the richest silver producing areas in the world back in the money.

The dumps contain cobalt, something the town was never particularly concerned with, despite its name. The stuff was considered somewhat of a nuisance and either piled up in unsightly dumps or was just left in pits, crosscuts, and drifts. But now J. J. Gray, president of Coballoy Mines Ltd., which has purchased and modernized the old Colonial mill and reopened the Trinova property, is about to give real meaning to the town's name. He estimates that more than $1.5 million worth of cobalt, along with silver and nickel, will be processed this year.

The town was originally named by Willet G. Miller, Ontario's first Government geologist, in 1903. It brought in more than $300 million worth of high grade silver from 1903 to 1928, some $100 million more than the total value of gold produced in the Klondike from the '89 gold rush to 1952.

(Continued on page 116)

Cobalt cont'd . . .

Rock dump discarded as useless by former silver miners in the Cobalt area may yield thousands of tons of cobalt-rich tailings. Dumps such as this one abound in the area.

Hand Sorting

The Trinova is reported to be the most valuable concentration of cobalt in the town. The mine was once known as the Nova Scotia. It had no mill on the property and ore was hoisted from the mine for sorting by unskilled, wasteful laborers. But silver veins stuck out of the rock "like the ribs of a broken umbrella." When the price of silver took a header, the mine was abandoned. Leftover cobalt ore is ripe for picking, above and below ground. Evidence of the unwitting extravagance of the old-timers is a pit containing about 30,000 tons of discards worth more than $22 per ton at current prices. Under the old hand-sort conditions, only rock with surface showings of high quality silver went to the mill. Some recent samples have shown as high as 20 pct cobalt, worth $700 per ton without milling.

Coballoy's Colonial mill is treating 100 tpd and making a 98 pct recovery of cobalt and silver. The mill's improvements were installed by Quebec Metallurgical Industries Ltd., which experimented with it and test-ran it for a year. The new Cobalt Chemical Refinery, owned by Quebec Metallurgical, utilizes a chemical-electro process, using Coballoy's mill products.

Nature Helps Out

The Coballoy mill is on a hillside offering gravity flow for ore entry into the mill and tailing exit into a lake. Peterson Lake, on the other side, is about as high as the mill entry, allowing for easy pumping of any quantity of water needed.

Nature supplies the compressed air used in the operation. Air is produced by sluicing water from the Montreal River into a shaft 350 ft deep. The long drop aerates the water running through a long tunnel with a high dome cut in its ceiling. As the water passes under the dome, the air escapes and compresses. It is fed from the dome through a pipeline at 120 psi.

Oil Rig Has Mine Exploration Possibilities

The worlds of the metal miner and the oil driller are rather separate and distinct, but the recent development of an oil derrick made completely of aluminum may find plenty of use in exploration for metals.

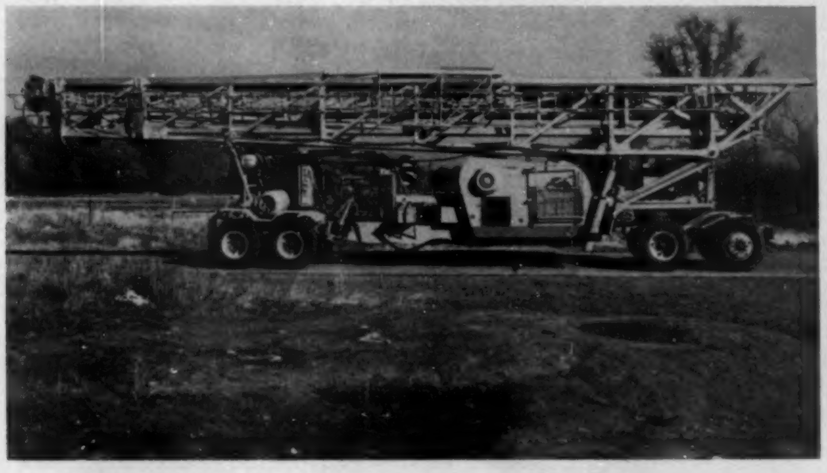
The first all aluminum oil derrick is in service in Oklahoma's Golden Trend oil field. The telescoping derrick, for use on a portable oil field servicing outfit, has square tubular aluminum legs and bracings to give maximum strength, while effecting a saving of 40 pct in weight.

Working together to build the rig were Reynolds Metals Co., Louisville, and Franks Mfg. Corp., Tulsa. They hail the derrick as the advance that puts the oil industry on wheels. In the past, it has been difficult to keep such equipment within highway weight limits.

The derrick extends to more than 100 ft, with 96 ft of free working space under the crown block. It telescopes like a fire engine ladder to less than 60 ft over the truck on which it is carried. The derrick is operated and the truck is propelled by a 300-hp power plant.

Derrick and unit are capable of servicing producing wells up to 12,000 ft and drilling to 5000 ft with 3½-in. drill pipe. The complete unit weighs 60,000 lb. Mast weight is 10,000 lb, a weight saving of about one third.

It may be quite possible to fly the rig into areas where drilling equipment has not been able to go up to now. Several mining men are speculating on possible exploration applications of the derrick.

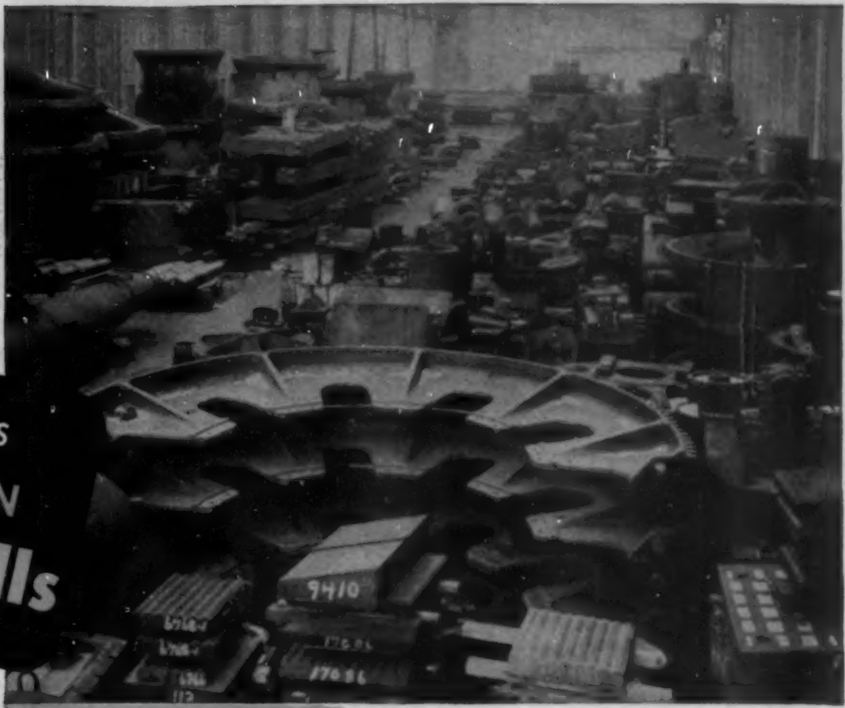


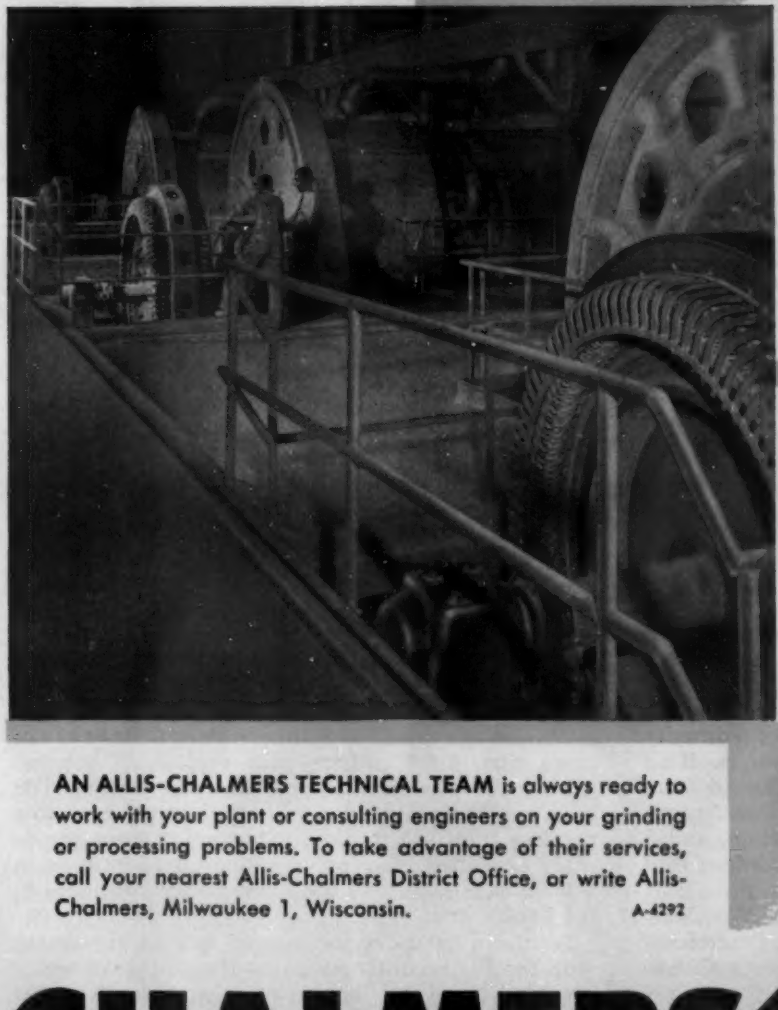
World's first oil derrick made entirely of aluminum, above, extends to more than 100 ft, with 96 ft of free working space. It telescopes and folds to less than 60 ft over truck. At left, the derrick is shown on portable oil field servicing unit.

Aluminum oil derrick being raised into position. Square tubular aluminum legs and bracings give maximum strength, yet make possible a 40 pct weight saving. Because of its light weight, rig may gain consideration in the metals exploration field. It can drill to 5000 ft.

TEAM

ALLIS-CHALMERS PROGRESS IN **Grinding Mills**





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A-C ALLIS-CHALMERS

GRANDMOTHERS, housewives, high school students, respected businessmen, and people of almost every kind of background you can think of are hitching their wagons to a Geiger counter. In Canada, it is becoming recognized that the amateur prospector is contributing a large share to the country's wealth. In 1954 more than 50,000 claims were recorded. Three fourths of them were registered by "week-end" prospectors. And there's still lots of room in Ontario. With every claim encompassing 40 acres, only 3000 of the province's 412,582 sq miles were staked. It's still possible for the amateur to jump in his car, stake his year's quota of 360 acres, and be back home by dark.

Part-time prospectors have suddenly found themselves shot into the millionaire class—but not too often. The know-how has got to be there—along with a liberal dose of plain luck. In Ontario anyone more than 18 years old can stake a claim on Crown land, on payment of a $5 miner's license fee. Mineral rights on much of the Crown or Government-held land granted after May 6, 1913, were retained by the Government. Claims must be registered within 30 days after staking. One prospector, so the story goes, staked a claim on a likely looking bit of ground and then never registered it, perhaps deciding it wasn't worth the trouble. Another group, who had lost out in staking the ground, although previously aware of the claim's value, waited. At the end of 30 days they registered the claim and were on their way to a fortune.

Schools have been attempting to fill the knowledge gap between desire to prospect and actually making a go of it in the field. At a two-week series of lectures last year more than 100 persons attended. And there were eight grandmothers registered for classes for prospectors held throughout Ontario last winter by the Government.

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THOMAS JEFFERSON WILLIAMS is a Georgia boy who lived in Tennessee and Louisiana, sold towels in Argentina, parlayed the receipts into an industrial empire, and is, as far as the Kappa Sigma Fraternity is concerned, Man of the Year.

Mr. Williams heads some 15 companies primarily engaged in the mining and chemical business. It all started in 1914 when Mr. Williams graduated from the University of Arkansas and decided that Argentina's air was for him. Affability, ambition, and a huge supply of towels from a Georgia cotton mill were his stock in trade.

"Son," the Georgia manufacturer told Mr. Williams, "I hate for you to stock up on this merchandise. Our experience is that you'll never be able to sell a towel below the equator."

Landing at Buenos Aires just before World War I began, he raised the price of towels in the Argentine and cabled home for more. Four decades later Mr. Williams is head of the organization whose most important mineral producer is Sociedad Minera Argentina S. A. During World War I, World War II, and the Korean War his mining enterprises in the Argentine produced vast amounts of tungsten, sulphur, and other chemicals. After visiting the U. S. recently, he returned to Argentina for the opening of a new concentrating plant at Los Condares tungsten mine, which he owns and operates. With a modern operation in terms of machinery and equipment, Mr. Williams feels that his advent into the concentrating end of the business is the only economic answer to the long sea hauls to consuming areas.

Mr. Williams has an abiding faith in the future demand for tungsten. "There will never be enough tungsten in this atomic age," he says. "The world supply of tungsten is a mystery. Tungsten is all underground. No one can guess the size of the deposits or the potential supply."

Among the honors conferred on Mr. Williams is that of Knight Commander of Daneborg presented personally by the King of Denmark.

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AMERICAN Smelting & Refining Co.'s announcement that it has completed preliminary negotiations for the financing of the Toquepala copper project in southern Peru is especially significant in the face of the growing tendency toward nationalism in South America. After only some mild research into the matter, one is left with the feeling that Peru is one of few South American nations where foreign capital still has that fresh, green, appealing look.

Under the agreement reached with Cerro de Pasco Corp., Newmont Mining Corp., and Phelps Dodge Corp., the Toquepala property, together with the Quellaveco property of AS&R, and the Cuajone property owned by Cerro de Pasco and Newmont will be transferred to a new corporation, Southern Peru Copper Corp. AS&R will own 57¾ pct of the capital stock; Cerro de Pasco, 16 pct; Newmont, 10¼ pct; and Phelps Dodge, 16 pct.

The Export-Import Bank of Washington announced last November that pursuant to the policy of intensifying its activities in financing economic development in this hemisphere, it approved in principle extension of a credit to Southern Peru Copper of not more than $100 million, plus capitalized interest during the construction and start-up period. The credit is contingent upon private interests investing not less than $95 million, inclusive of sums previously expended. This private investment must be in a form satisfactory to the bank and subordinate to the bank's loan.

Southern Peru Copper entered into an agreement with the Peruvian Government Nov. 10, 1954 which provided for stable income rates, waiver of import duties, freedom of copper exportation, and freedom of exchange. It has been reported that Southern Peru Copper has agreed to start operations within 18 months. In return, it is said, the Government

agreed to impose taxes of not less than 10 pct and not more than 20 pct on the profits resulting from exploitation of Toquepala and nearby Quellaveco.

Drilling of the Toquepala property was completed in 1952, with an ore reserve of more than 400 million tons with an assay slightly greater than 1 pct, proved. Southern Peru Copper now has to get about the business of preparing the deposit for an open pit operation turning out some 30,000 tpd. A townsite, concentrating plant and other facilities near the mine have to be built. Plans call for a standard gage railroad from the mine to the seacoast at Ilo, Peru, about 110 miles away. Smelter, power plant, and townsite will be constructed at Ilo, along with port works, warehouses, and other facilities. It will take about five years to get the job done at an estimated cost of about $200 million.

The Quellaveco and Cuajone properties also contain large porphyry-type copper deposits. Both have been drilled. Their location makes it possible to eventually use many of the major facilities planned for Toquepala.

It is expected that the project will revitalize all of southern Peru, stabilize the *sol*, give employment to thousands, and lead to the creation of other industrial projects.

Peru's welcome mat for foreign investment seems to be paying off, particularly in respect to the U. S. There are no restrictions on the transfer of earnings or repatriation of capital. Right now large projects under construction include a steel plant at Chimbote, the vast Paucartambo hydro-electric development, oil exploration, and a road building and colonization program in the forest area near Pucallpa. Morrison-Knudsen Co. recently signed a contract with the Government to complete the Chotano-Chancay river diversion tunnel in northern Peru.

AS&R is not saying how much it has already put into Toquepala. The nearest one can come to some kind of figure is the statement in an annual report that $8 million had been invested by the end of 1953.

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IT has become part of American tradition for big company representatives to descend on college campuses every June in an all-out attempt to grab engineering talent before the talent goes into the market place. There can be little doubt that the senior engineering student is aware of his importance—15 columns of engineering help wanted advertisements in the classified section of the New York Times every Sunday attest to the demand.

The engineer-to-be is usually given the full treatment. By the time the company representative is finished the prospect is certain that he is the last hope of industry; fame and fortune are his for the asking, and the nation has been stagnant, awaiting his particular genius to move it to the heights. Following graduation comes disillusionment.

The engineer may discover that his function is little understood by public, employer, or worse, by himself. There are several definitions for the engineer, according to George S. Odiorne, writing in a recent issue of *Harper's Magazine*. The society sees the engineer as someone who has accumulated technical knowledge above the workman stage. Salesmen and accountants could fit that definition too. The theoretical scientist pictures the engineer as a "watered-down physicist, chemist, or mathematician; a sort of hairy-chested scientist-gone-wrong." The engineer also has an idea of his own identity. Some still view themselves as Thorstein Veblen, the engineering-minded sociologist of the century's early years, pictured them—tough, neutral technicians—the best qualified to manage society. However, many pure scientists today view the engineer as someone to convert the products of the laboratory into practical applications and the best engineer as the one "most thoroughly steeped in science."

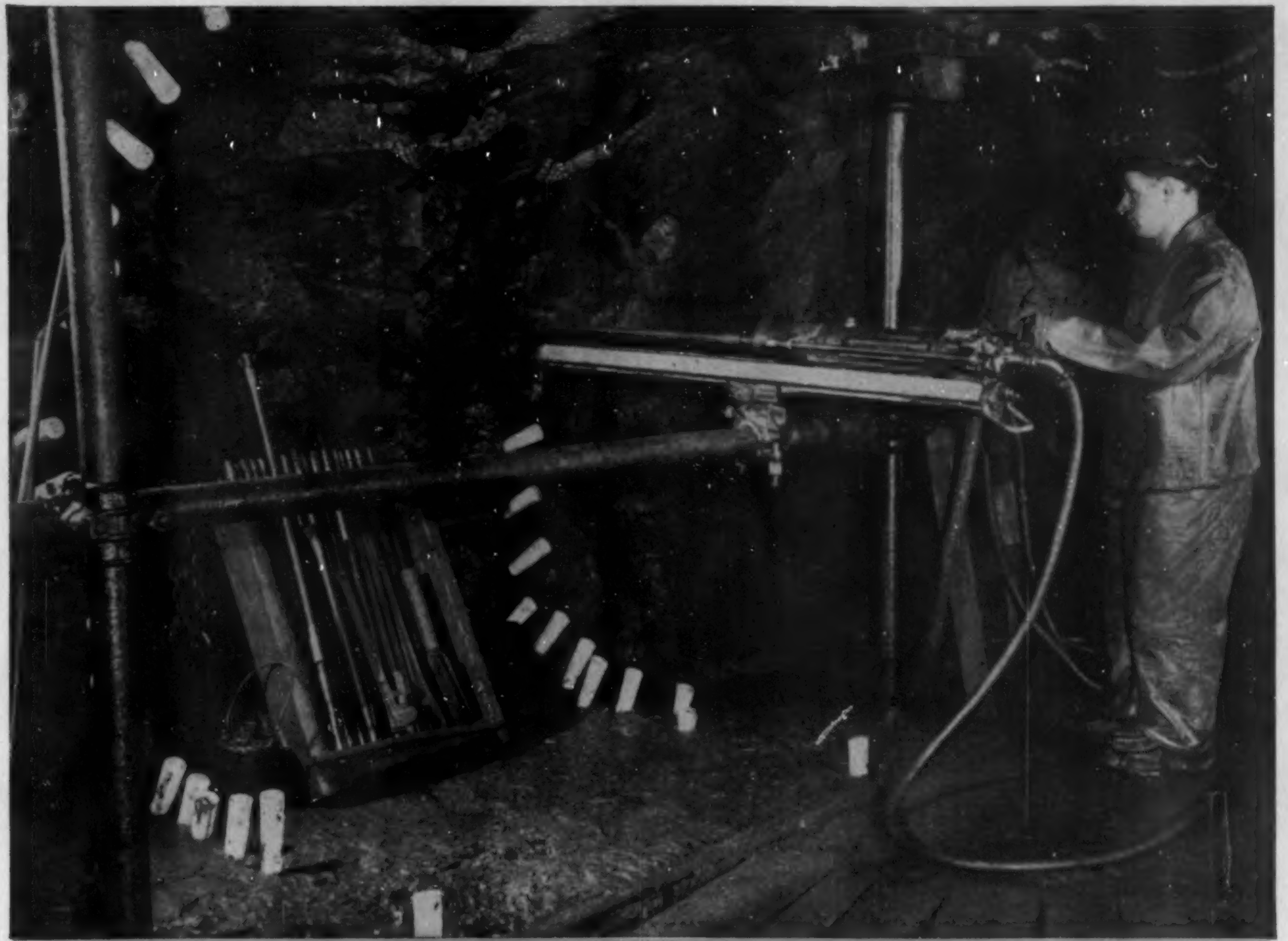
Some engineers, according to Mr. Odiorne, are half convinced of the validity of the idea, and thus seek to isolate themselves from the other end of the scale, where one finds the psychologists, sales managers, and public relations counselors. But the engineer also finds himself rejected by the pure scientist on one hand and often by the pure businessman as well. "He is by no means as lavishly paid as his scarcity might lead you to suppose, and even within industries of the highly technical nature his prestige may be far lower than that of the most junior executive," Mr. Odiorne states.

A *Fortune* article quoting the vice president of an electrical company supports Mr. Odiorne's stand; after pointing out that the ratio of engineers to non-engineers had risen.

"This results not from an increase in the proportion of engineering work but rather from the fact that engineers, as relatively cheap help, are now used for so many other activities than those for which they were trained. Anyhow, there is a general assumption that you can make more money being a manager. This has gone so far in my company that all engineers in supervisory positions are designated 'managers of engineering.'"

Mr. Odiorne traces much of the engineer's trouble to an overall failure to define "what an engineer is," other than to call him a graduate of an engineering college. He notes that "To their credit, many thoughtful engineers recognize" that a certain anxiety concerning the idea that the engineer is poised and self sufficient—balanced between two chasms into which he must never fall—underlies overt complaints about salary, job definition, and a higher place on the industrial scale. He feels that much of the job watering down (Mr. Odiorne calls it blending) has been brought about by engineering initiative. He favors this blending, but feels that management must also look to the best use of engineering talent.

M. A. Matzkin



FOR A RECORD BLAST—

explosives research pays off

A hill of hard, seamy granite 65 feet high, 100 feet wide, and 600 feet long, located between two sections of a new highway near South Billerica, Massachusetts, was broken to grade level on August 19, 1954, by the largest single blast ever made on a New England highway construction project.

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XR55-1

Drift of Things

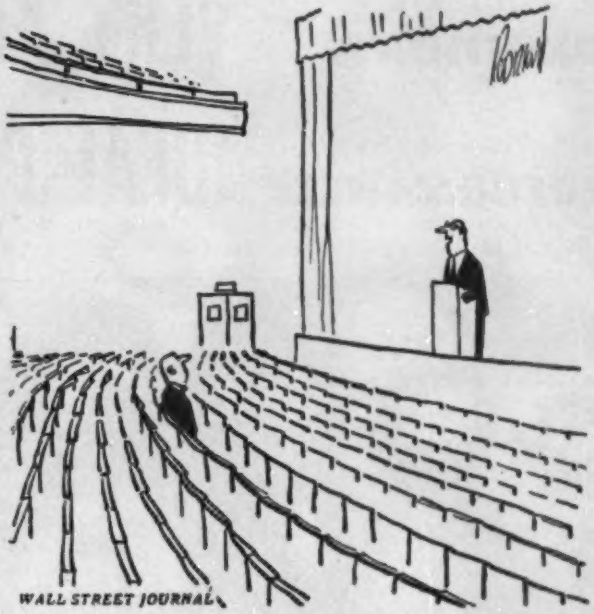
SEVEN AIME members are going to be annoyed, even angry, and the frustrating part is that there is nothing we can do about it. There isn't when there is no name or address on a Free Literature reply card. When people mark items, tear out the card, and mail it we want to give them service. So do the companies whose products and bulletins are mentioned. But when there is no address on the card that's it. You can't even apologize. One or two blank cards a month is about par, but the only way we can account of a small landslide of seven blank cards this month is the holidays. At least all the "no address" cards are postmarked around Thanksgiving and Christmas. Try again. Thanks.

CURRENTLY going the rounds is the following: *Blessed are those who go around in circles, for they shall be called big wheels.*

A RECENT report from the Utah Section outlined steps being taken to foster student participation. Among the things done was the selection of one student by each section member, a sort of sponsorship at the section doings which helped make the students feel that they belonged and were not strangers. Cost of the tab at some section meetings is sizable in student eyes, and several sections help out on this, some with a lower rate for students attending certain meetings, with the difference being absorbed by a fund for the purpose.

It is worth-while to mention what the San Francisco Section has worked out with the two nearby student groups, one in Berkeley at the University of California, and the other at Stanford University in Palo Alto. Each year one of the student sections is host to a combined meeting of the parent section and the other student group. Students take responsibility for programs, entertainment, and the host student president acts as chairman. This sort of thing may be done quite widely, frankly we don't know, but if it isn't in use the idea is free and sounds fine. Among the benefits, aside from a vacation for the harried program chairman, are: a good opportunity for both grads and friends to get posted on the present state of facilities and faculty; an opportunity for the student group to show what it can do; a greater sense of student participation in section and AIME activity; and lastly a fine afternoon or evening program.

COULD the State Department have made a mistake in barring Soviet citizens from Brooklyn? Let us suppose that some Russian in search of amusement wanders across the Brooklyn Bridge and happens upon Ebbets Field. He plunks down a few rubles and enters into this particular area of summer madness. Our Russian friend suddenly feels at home. Them bums are playing Cincinnati. But he makes a mistake. With a daring that comes only from ignorance he cheers the Reds. This could be fatal. However, there are rational people in Brooklyn who never talk louder than five octaves above a boiler works. With one thing and another and the way arguments go in Flatbush, he is by now involved in international politics. And no one is a greater authority—on politics or anything else—than a Brooklynite, whose knowledge of baseball, hot dogs, and what will happen to those unspeakable Giants next year can be matched by no mortal.



"... some of us would view the subject '*resiliency of painted metals*' as one lacking of interest ..."

Our Russian cannot withstand the onslaught of fact brought to bear. In no time he is a counter-revolutionary, an undying worshiper of Gil Hodges, and a hater of authority—umpires, dictators, or commissars.—M.A.M.

ECONOMIC barometers and forecasts of things to come are a dime a dozen any year. There is one price indicator that seems to make a lot more sense than those put out by the banking houses and Government. We refer to the semi-annual 2-in. thick volumes of help and inspiration put out by Sears and Wards. The spring bibles are at hand across the nation, and the statistically minded report that catalog prices are down 2½ to 3½ pct in most mail-order house lists. Regardless of what happens to production in 1955, and the prophets are throwing roses around, we will take the mail-order people's word anytime on prices. And they say down.

Further research in Sears this year shows that Geiger and scintillation type counters now rate a full page. Prices range from $19.95 to $995.00, under a banner head, FIND URANIUM. It has been duly reported by historians of our times that the listing by Sears and Wards of equipment for inside toilets had a lot to do with changing the habits of the nation, and these same merchants had the foresight to stock automobiles when "Git a horse!" applied to something other than a slow jockey. We won't force the issue on the counters, but if it can be done those mail-order boys will have a mine in everybody's backyard.

R. A. Beals

Will Taxation Destroy the Mining Industry In Mexico?

by Rixford A. Beals, Associate Editor

IT has been observed with considerable accuracy that if the major U. S. mines were moved to Mexico tomorrow, many of them would be losing money. But the companies would continue to operate, losses or not, because they would have to. And some of the newest and most promising low grade ore projects in the U. S. would never have "gotten off the ground" had they faced the equivalent of the mining taxes now in effect in Mexico.

No one of the various mining taxes in Mexico seems prohibitive by itself. Taken together, on the balance sheet, however, the picture changes. Production and export taxes alone take from 20 to 40 pct of the gross product value for base and precious metals. This is a percentage *before* deduction of any mining or treatment costs, or amortization, and is based on the crude metal content. On top of this there are income, excess profit, and dividend taxes, which in total closely approach the present U. S. corporate income tax level.

It might be pointed out that these various taxes do not single out the foreign-owned property specifically—they apply to Mexican and foreign owners alike. In hardship cases, remission of a part of the taxes is a possibility, but that is a matter of extensive negotiation.

Largest burdens arise from the two taxes based directly on gross metal output: the production tax and the ad valorem or export tax. Local operators aptly term them *absolute* taxes. The production tax ranges from 2.4 pct for zinc up to 20.1 pct for gold, while the additional export tax is 25.5 pct or more for all major nonferrous metals and silver. Combined, the taxes range from 20.1 pct for gold to 38.9 pct for silver. These figures are as of January 1955. Exact rates for the various taxes vary from month to month as metal prices change. In fact it is difficult if not impossible for a company to predict what absolute taxes it will be paying in the immediate future, since the exact rate of taxation will depend not only upon metal prices, but also upon the varying proportion of the various metals in the ore. In the case of the complex base and precious metal ores common to Mexico, this presents real management problems. New mines, reopened mines, and small-scale producers obtain some relief from these tax rates in the form of subsidy.

Over and above the taxes on output there is an income tax ranging up to 35 pct on income over $125,000,* an excess profits tax of 5 to 25 pct, and a dividend and undistributed profits tax, payable regardless of whether or not dividends are declared, amounting to 15 pct. Finally there are provisions in the tax law for depreciation, but none for depletion. Thirty pct of the undistributed profits tax may be deferred by investing this amount in Mexico.

* All figures in this article are given in U. S. dollars, figured at the present rate of exchange, approximately 12.50:1.

The most outspoken criticism of present mining taxation policy in Mexico has come from Mexican citizens themselves, including members of the present and previous governments. Writing in responsible publications these men have been vehement in holding that present taxes are causing long-range damage to the mining industry in order to provide immediate revenue.

In 1945, Constantino Perez Duarte, more recently Assistant Minister of Economy, expressed these ideas in his article *Mexican Mining Must Have Tax Relief.*'

As part of the program with regard to mine taxes, all those which are a burden to the industry should be studied, especially the following:

1. To repeal production taxes as they are obsolete.
2. To repeal the *aforo* or export tax because the reason for which it was established no longer exists.
3. To make a careful study of the income tax in order to make its application simple and equitable.

Investors must enjoy reasonable protection and must be given a chance to earn profits commensurate with risks taken.

How Absolute Taxation Could Reduce Ore Value*	U.S.	Mexico
Gross Value	$800,000	$800,000
Treatment	200,000	200,000
Production and Export Tax**	none	280,000
Net Smelter Value	600,000	320,000
Mining Cost***	500,000	300,000
Operating Profit	100,000	20,000
Depletion	50,000	none
Net Taxable Income	50,000	20,000
Tax (approximate)	25,000	10,000
Net Income	25,000	10,000
Dividend Tax	none	1,500
Net Cash Gain	75,000	8,500
Government Cash Gain	25,000	298,500
Ratio: Company to Government Gain	1: 0.33	1: 35

* These hypothetical figures for a complex Pb-Zn ore were derived from study of tax schedules and company reports.
** Basis, average 35 pct overall tax.
*** Base 50,000 tons; U.S. at $10 per ton; Mexico at $6 per ton. Difference in costs may be less.

MEXICO	UNITED STATES
net profit 2 pct	12 pct net profit
operating costs 41 pct	56 pct operating costs
taxes 37 pct	12 pct taxes
freight and treatment 20 pct	20 pct freight and treatment

Figures for base metal mine in Mexico, 1954, and figures for the same operation if it were in the U.S., using the same cost ratio as on p. 127.

According to a recent article,[8] the Mexican Minister of Economy supported the suggestion that capital investment in low grade metal deposits be encouraged by applying only the income tax. Even such an approach raises the question of defining low grade ore.

In 1954 the taxation discussed nine years earlier by Duarte had become far more severe. We quote remarks of C. E. Temperley, chairman of the board of San Francisco Mines de Mexico, as reported by *The Financial Times* (London). In his summary at the 41st annual meeting of the corporation he pointed out:

"If metal prices were to endure at the level ruling in the first quarter of 1954, which was a very low level indeed, and if other conditions were the same as those then ruling, your Corporation would earn little or no profit available for dividend but it would still have to pay the Mexican Government more than $4 millions a year in taxes on the gross value of its products! And this dreadful toll would be increased to $5 million a year by the recent increase in Export taxes . . .

"American firms have been rather reticent about the matter of mining taxation in Mexico, and understandably. Investments are high, it is a matter of delicacy not to criticize your host, and the situation could become worse. How much worse, before the companies become vocal, is a question. For example the last financial report of the Fresnillo Co. states:

"Operations for the year under review were again adversely affected by low base metal prices, although some slight improvement was experienced toward the end of the period [June 30, 1954]. Taxes again took a very large proportion of the Company's revenue and this proportion was increased by a rise in the rate of export tax imposed by the Mexican Government, following upon devaluation of the peso in April 1954. This devaluation resulted in a dollar exchange loss of $202,397. As a result of the rise in the internal price level after the devaluation, a general wage increase of 12 pct became effective on May 15, 1954."

In evaluating the tone of the quotation above it is helpful to have these figures from the Fresnillo report for the year ending June 30, 1954:

Gross Revenue from Metals and Ores produced	$17,965,634
Operating, Shipping, Selling, and General Expense	16,490,884*
Mining gross income	1,474,750
Mexican Taxes on Income	482,463**
Mining Income before deduction for depreciation, development, etc.	$ 992,287
Net profit	306,984

* Included are $4,700,000 in production and export taxes, and total Mexican taxes paid in the fiscal year amounted to about $5,200,000 or 29 pct of gross revenue.

** Including income, excess profits and dividend taxes.

More recently, in January 1955,[9] we have this expression of the situation, again from within Mexico:

Labor in Mexico

The Mexican miner is well protected, both by provisions in the Mexican Constitution and the specific collective contracts that exist between the individual mines and the labor unions. By carefully worked-out schedules based on length of service and rates of pay, miners are compensated for accident and their families for death. For example, the family of a miner killed accidentally is given three years salary. Incapacitation due to accident or silicosis is worked out on a percentage basis, the percentage being decided by examinations conducted by mine and union doctors. Surface and underground men are given full retirement pay after 20 and 15 years of service, respectively.

From the point of view of the operator the right to fire a miner is an objectionable feature of most collective contracts, since the law confines such right to cases where an overt act is committed. Should a company take the step of firing a miner, immediate appeal is invariably exercised, and this process can be long, expensive, and indecisive. If a man quits of his own volition there is no severance pay involved, but such cases are unusual. Thoughtful Mexicans have advocated a system comparable to our own Social Security. This, of course, would presuppose that the Mexican Government would set up the necessary machinery.

In all matters of contention between operators and labor unions there is a labor board in the local area, consisting of Government, union, and mine representatives. The decision of this board can be referred to Mexico City for review by the Federal Labor Board there, with final appeal to the Supreme Court.

"Of the approximately 3400 mining properties in the Guanajuato and De La Luz mining zones, only 26 are currently being worked, 300 are idle though their concessionaires hold onto their rights, while 3075 have been abandoned or turned over to the national reserve, reports the National Chamber of Commerce of Guanajuato City. The organization blames the situation on excessive taxation, and heavy indemnization payments to employees which, it charged, caused the collapse of the *Guanajuato Reduction & Mines Company* and the *Guanajuato Consolidated Mining & Milling Company*. The chamber pointed out that taxes on gold and silver production rose from 2.5 pct in 1905 to 19.9078 pct in the case of gold, and to 35.16 pesos per kilo in the case of silver, plus 25.5 pct of the present silver quotation of 340.86 pesos a kilo. In addition to these levies on gross production, the chamber added that mining companies must also pay taxes on income, a special mining tax, and others for schools, weights and measures, inspection of motors, as well as a State predial impost. As a possible solution, the group urges a program designed to rescue mining from what it calls 'complete decadence.' It calls for abolition of all taxes on the industry with the exception of the income tax, temporary exemptions from import duties on machinery and equipment essential to the industry, and the turning over to the Government social security agency of the obligations of indemnity payments to employees retired for incapacity."

The Effect of Devaluation

Currency devaluation has been part of recent economic trends, and Mexico has not escaped it. As recently as seven years ago the peso was at 4.85 to the U. S. dollar; today it stands at 12.50 to 1. The effect of the last devaluation in 1954 on the operations of the Fresnillo Co. is clearly noted above. To cite another example, here are further remarks from C. E. Temperley of San Francisco Mines de Mexico:[4]

"Our metals are sold largely for dollars and quite a substantial proportion of our costs are in pesos so that if nothing else had been done, this devaluation (from 8.65 to 12.50 pesos to the dollar) whatever its other virtues or failings turn out to be would have been of some benefit to the hard-pressed mining industry. But, alas, at the same time that devaluation was announced, the Export tax on metals was raised from 17.8 to 28 pct. The effect of this is that the major part of any possible benefit . . . has been appropriated by the Government at the source. The small benefit which remains is highly vulnerable. Internal costs of all sorts tend to rise after a devaluation and they will not have to rise very far in Mexico to make the mines there definitely worse off than before."

The matter of internal costs brought out by Mr. Temperley has been a very real problem. For Mexico has not escaped inflation either. In fact the percentage rise in staples at the lowest cost-of-living level has been high. This has been understandably reflected in a series of wage demands which had little relation to metals prices.

How Much Lower are Mexican Mining Costs?

Many of the older American engineers may still have in mind mining costs in Mexico before World War I, or at least of the period before 1940. The situation, as suggested by devaluation and inflation, has altered radically since then. Although it cannot be denied that hourly wages are lower in Mexico than in the U. S., materials and equipment costs are higher in Mexico. Higher costs are due to import duties, increased freight, peculiarities of locally available items, and necessity in most cases for larger inventory. The difference in overall costs between the two countries is less than might be expected.

Mining in Mexico:

Production, wages, costs, and taxes for the major mining companies in 1952. (Last year for which complete figures were available.)

Production

Gold	Kilos*	6,376
Silver	do	794,586
Lead	Metric Tons**	182,304
Copper	do	42,600
Zinc	do	200,121
Prepared Coal	do	551,483
Coke	do	350,580
Coke Breeze	do	22,608
Pitch	do	9,492
Sulphate of Ammonia	do	3,247
Sulphuric Acid	do	35,618
Zinc Sulphate	do	1,242
Benzol	Liters	3,516,943
Creosote	do	5,538,095

Number of Men Employed	29,492
Total Wages and Salaries (including indemnities)	$32,600,000

Taxes

Production	$18,728,000
Export and Ad Valorem	27,399,000
Income, Dividend, and Excess Profits	26,821,000
Other Taxes	2,911,000
TOTAL	$ 75,895,000

Supplies Purchased in Mexico (including power)	$23,000,000
Freight and Express	9,250,000
Royalties	1,270,000
Miscellaneous	12,370,000
TOTAL	$ 45,955,000

Total Left in Mexico	$154,450,000
Supplies Purchased Abroad	$ 16,744,000

* Approx. 2.2 lb. ** A long ton, approx.

From first-hand contact with labor in Mexico, verified by the experience of others, the author can state the Mexican miner is carefully—even rigidly—protected by both labor laws and union contracts. Typical union contracts not only cover conditions of employment, but compensation for injury, retirement, and separation pay in the case of layoff. Discharge of a worker may be only for commission of an overt act, and then may be appealed by the worker or the union as far as the National courts. The right to strike is well insured.

No one criticizes the individual Mexican miner's ability, but the cost factor of production efficiency is not necessarily related to this. Efficiency will be low if orebodies are complex, workings are old, equipment is dated, or mechanization lacking—and all these factors are to some degree present in Mexican mines. Lack of available capital and of incentive to bring in new capital has tested the ingenuity of many an operator and has lowered production efficiency, without regard to the miner's actual work.

One Bright Spot

Small-scale mining operations get some tax relief in the form of subsidies, particularly properties producing no more than a few hundred tons of ore per month. Relative importance of this program can be judged by these figures: In 1954 the estimated return to the Government from production taxes was given as about $60 million, while for the same period the discount as subsidy to small mines was about $2 million. The latter figure was quoted as applying to mines where production did not exceed $16,000, and many of the mines receiving this aid fall far below this figure. Critics have pointed out that even if the smaller properties are preserved within the limits of the program, it falls short of providing incentive for capital investment or opportunity for capital growth that will create great mines from these small properties. In fact, under these conditions the temptation is to keep production down and artificially create small production units.

Why then the continuing interest in new mines in Mexico, as reported in the press? The answer is twofold. Taxation, now striking hardest at the nonferrous and precious metals producers, is scaled down for industrial minerals. Secondly, there are tax subsidies for new mines during their first few years of operation, and for reopened mines which had been unproductive for ten years or more. These two points, coupled with relief for the small-scale operation, explain the pattern—small, new properties, especially in the industrial minerals field.

Apparently even the men who have had a major part in framing the tax laws do not like them. But the tax burden has not been placed capriciously on mining in Mexico. The Government is faced with the absolute necessity of raising sufficient capital to carry out an ambitious program of industrialization and power and irrigation projects. The situation should not be blamed on any one person or group—it was originated by necessity and has continued and worsened. The time may be coming, however, when the danger of permanent damage to the national mining industry will dictate that relief be granted to this segment of the economy and that adjustments be made in other directions.

Faced with pressure from two directions—rising taxes and constantly rising wages—the operators have few choices remaining. Those with high grade properties continue, and those with medium grade ore are forced to mine their better orebodies and endanger lower grade reserves. Those with high costs or low grade ore go under or survive through subsidies.

Some close observers of the scene feel that the present absolute taxes cannot as a practical matter be eliminated immediately. They hope, however, that some means will be found for reducing and eventually eliminating the absolute taxes. Such action, they argue, would reassure those operators now in Mexico, and would lead to an influx of capital such as has resulted from Peru's revised mining tax approach and from Canada's policy of encouraging new operations by tax exemptions. The widened tax base would then result in greater tax revenue from mining, increase employment, and create a better economic climate.

References

[1] Constantino Perez Duarte; Mexican Mining Must Have Tax Relief, *E&MJ*, December 1945.

[2] C. W. Wright: What Chance Has Foreign Capital in Mexico, *E&MJ*, March 1954.

[3] Report of the National Chamber of Commerce of Guanajuato, Mexico.

[4] *The Financial Times*, (London), May 19, 1954.

Mining Laws and Taxes in Mexico

1. **Rights of foreigners.** Under Mexican law foreigners are allowed to engage in mining but must waive their rights to diplomatic protection. This law is comparatively recent and does not affect the older companies.

2. **Absolute taxes.** Each month the Minister of Finance in Mexico City publishes a schedule of taxes which varies with the prices of metals. At the moment the following schedule is in effect, based on percentage of smelter rates.

(Jan. 1955)	Production tax	Export tax*	Total pct
Copper	4.3	28.5	32.8
Lead	6.0	28.5	34.5
Zinc	2.4	28.5	30.9
Silver	13.4	28.5	38.9
Gold	20.1	exempt	20.1

* Includes ad valorem export duty, which ranges up to 7 pct.

New mines or mines which have not been worked for over ten years obtain some tax concession for five years ranging from 10 to 50 pct, the amount determined by direct negotiation with the Minister of Finance (Hacienda).

Small mines shipping up to 250 tons of ore per month obtain reductions of from 30 to 75 pct of absolute taxes. Mills treating a maximum of 10 tpd can obtain a 75 pct reduction on absolute taxes. On a 50-ton daily schedule, the tax concession may be 50 pct.

In 1953 absolute taxation was revised to assist small mines. Refunds of up to 75 pct are now possible for mines paying absolute taxes of less than $16,000 per month.

3. **Dividend tax.** There is a 15 pct dividend tax payable whether or not distributable profits are paid to stockholders.

4. **Pertinencia taxes.** The Mexican Government does not grant surface rights to holders of pertinencias. These rights must be purchased from the individuals who may be residing or working on the land in question. A pertinencia, equivalent to 2½ acres, carries an annual tax of 6 to 15 pesos a pertinencia, the smaller amount applying to a holding of 5 pertinencias or less and the larger amount to 100 pertinencias or more.

5. **Transfer of capital.** There are no restrictions on transfer of capital or conversion of funds.

6. **Income taxes.** Income taxes range from 4 pct on income up to $160 to a maximum of 35 pct on an income of $80,000.

7. **Excess profits tax.** There is an excess profits tax ranging from 5 to 25 pct on profits exceeding 15 pct of the working capital.

8. **Amortization.** The tax law permits 5 pct amortization on real estate and other immovable property investments but this can be increased to 10 pct in special cases. Likewise depreciation of 10 pct on movable property investment is permitted and this figure may be increased to 20 pct under special circumstances.

9. **Ore depletion.** There is no allowance given to ore depletion. The Mexican Government takes the position that all ores belong to it until such time as it is removed from underground and shipped to a smelter or placed in a beneficiating plant.

10. **Subsidies.** When an individual mining enterprise can show it is impossible to operate profitably, application for a subsidy can be made. This takes the form of reduction in absolute taxes.

11. **Abandonment of property.** No mine can be abandoned without prior consent from the Mexican Government.

12. **Indemnity payments.** In the event permission is granted to shut down a property, indemnity payments to workers are then in order, these being calculated on the basis of prevailing wage scales and length of employment.

Choosing Ore Feeders For Beneficiation Plants

by O. W. Walvoord

A Definition:

An ore feeder is a mechanical device that, by virtue of its motion, causes ore to be supplied or carried forward at a desired metered rate to other milling equipment.

FROM the viewpoint of economics there can be little question, even for small tonnage operations, as to the advisability of using mechanical feeders in the place of manual labor. Only a few short years ago, it was common practice to employ *crusher men* for the feeding of crushers, thus eliminating the capital expenditures required for mechanical feeding equipment. Having been a crusher man in several instances, the author does not favor such practice. The high cost of labor today eliminates the crusher man, and feeders become an economical capital expenditure when considered against the importance of obtaining maximum production efficiency from all the machines in a process flowsheet.

Accurate feeder control is of major significance in obtaining any desired degree of metallurgical efficiency. By the same token, accuracy of a full-scale mill metallurgical test is only as good as the accuracy of the feeder performance during the test. It is probable that many mill tests of the past have been inaccurate because the accuracy of the feeders was below the calculated accuracy of the test.

Many feeders have been casually and erroneously chosen because of lack of compiled up-to-date information on the subject. With an ore of constant character, unvarying assay, sizing, consistency, and moisture content, the choice of a feeder is not too difficult. But it is a rare mill where these factors do not vary in the space of a few hours. Introduction of these variables complicates the problem, and the wider their fluctuations, the more difficult becomes the choice of the feeder.

Bins as Related to Feeders

Broadly speaking, the bin is a reservoir for the ore, from which the feeder draws its supply. But the feeder cannot function as such if the ore does not reach the feeder. If the ore is free flowing then there is no problem. Where there is only a small tendency of the material to *hang up*, this may be largely overcome by increasing the angle of slope of the bin bottom so that it exceeds the angle of repose of the material. In square or rectangular bins, the valley angle is the critical slope, as it will be flatter than either of the sloping bin sides whose intersection form this valley. Various manufacturers' catalogs include complete valley angle charts; the following is a tabulation of a few bin slopes and valley angles:

Bin Slope	Valley Angle
45°	35°
55°	45°
60°	50°

For the more sticky ores, where the tendency is to pack, hang up, and bridge over the opening, the problem is more difficult. Here the mining practice of keeping the total column of ore moving has been applied, which explains why bins with multiple openings are preferable to those with single openings for handling sticky ores. Multiple bin openings also minimize ore segregation. With many ores the degree of stickiness increases as the percentage of fine material increases, making it advisable to provide bin storage for relatively large size ore only and to perform fine crushing as a continuous process without fine ore storage.

Small tonnage operations: Where capital investment must be held to a minimum, it is suggested that sticky ores be handled with feeders, the area of which represents a large percentage of the area of the bottom of the bin. In other words, a "moving bin bottom" is a fairly positive method of sticky ore removal. Bins 1, 3, and 7 of the chart on p. 134 show how bin discharge openings can be designed to approach a "moving bin bottom" and thus reduce the arching effect of sticky ores.

Large tonnage operations: It is questionable that any of the above suggestions are economically practical for very sticky ores. The author believes it unwise to recommend any feeder for handling large tonnages of very sticky ores. Rather, the problem should be approached from a different viewpoint, and the ore first treated to eliminate or neutralize its sticky character. This can be done by desliming, or by converting the material to an all wet or all dry condition. Though this adds to capital and operating expense, it is still more feasible economically where top production must be maintained.

One particularly difficult problem is the handling and feeding of concentrates. The general trend in recent years has been to avoid feeders for this operation and to use front-end loaders.

(Selection Chart on pp. 132-3; text continues on p. 134.)

O. W. WALVOORD is President of O. W. Walvoord Inc., Denver.

Ore Feeder Selection Chart

The Ore — The Job — The Cost

Here are all the types of feeders most commonly used in beneficiation plants as well as a review of the various factors that play a part in the choice of a feeder.

A—Ore Characteristics

1)—*Size*—This is normally described as maximum particle size.

2)—*Mobility*—This is probably the most important single factor governing ore feeder selection and operation. Various classifications of mobility are:

a) *Free flowing ores* can be described as those ores whose mobility resembles loose gravel, and which readily flow when the normal angle of repose is exceeded.

b) *Flooding ores* are those fine ores that even when powder dry have a tendency to act as a liquid. Examples are dry cement and barite.

c) *Slurries* are solids mixed with an excess of water. Another common term is *mill pulp.*

d) *Tendency to stick ores* are those that can be classified between free flowing ores and sticky ores. Perhaps a more descriptive term is *sluggish ores.*

e) *Sticky ores* are those that will stick-up and be nonmoving en masse, as well as those in which the fines adhere to the bin and hopper sides and have a tendency to build up. The bridging effect of this type of ore is often greater than the minimum hopper throat opening or bin discharge opening shown in the chart, upper right, p. 133.

The major ore feeding problem is definitely the handling of these sticky ores. When this is encountered, it may be more economical, as mentioned before, to go to an all wet or all dry process, or wash and deslime the ore. Also, storage of coarse crushed product, only, is a possibility.

3)—*Miscellaneous:*

a) *Abrasive action* is important from a maintenance standpoint.

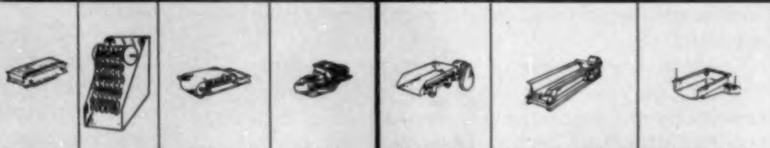
b) *High temperatures* can be detrimental to some feeders and the manufacturer of the feeder should be consulted.

B—Feeder Functions

1)—*Uniformity of Discharge*—Feeder discharges vary from a uniform stream, such as from a conveyor belt, to a spotty or slug discharge such as comes from a chop or reciprocating plate feeder.

2)—*Accuracy of Weight of Discharge*—This is an important factor in most installations. Ore feeders can be divided into two classifications:

Selection Chart for Ore Feeders in Beneficiation Plants



		FEEDER →	CHAIN AND BELT				TROUGH		
			STEEL APRON	CHAIN	BELT (VOLUME)	BELT (WEIGHT)	RECIPRO-CATING	OSCILLATING MECHANICAL	VIBRATING ELECTRICAL
ORE CHARACTERISTICS	SIZE	COARSE ORE, 48 TO 0 IN.	√	√					
		COARSE ORE, 12 TO 0 IN.	√	√			√		√
		MEDIUM ORE, 3 TO 0 IN.	√	√	√	√	√	√	√
		FINE ORE, $\frac{3}{4}$ TO 0 IN.	√		√	√	√	√	√
		FINE ORE, 10 MESH TO 0 IN.	√		√	√	√	√	√
	MOBILITY	FREE FLOWING ORES	√	√	√	√	√	√	√
		FLOODING ORES							
		SLURRIES							
		TENDENCY TO STICK ORES	√		√	√			
		STICKY ORES	√		√				
	MISC	HIGH TEMPERATURES	√				√	√	√
		ABRASIVE ACTION	AVG	LOW	LOW	LOW	AVG	AVG	LOW
FEEDER FUNCTIONS		UNIFORMITY OF DISCHARGE	GOOD	GOOD	GOOD	*GOOD*	AVG	AVG	AVG
		ACCURACY OF WEIGHT OF DISCHARGE	AVG	AVG	AVG	*GOOD*	AVG	AVG	AVG
		RANGE OF FEED REGULATION	GOOD	AVG	GOOD	GOOD	AVG	AVG	*GOOD*
COSTS		CAPITAL INVESTMENT	HIGH	AVG	AVG	AVG	LOW	AVG	AVG
		MAINTENANCE	HIGH	*LOW*	LOW	AVG	LOW	LOW	LOW
		POWER REQUIREMENTS	HIGH	LOW	LOW	LOW	AVG	LOW	*LOW*

(Feeder Functions continued)

a) *Volumetric Feeders:* This includes by far the largest number of feeder installations. The volume of material fed by this type of feeder can be automatically controlled by weighing devices, such as conveyor scales, thus converting to constant weight per period of time. The large variation in weight of material per unit of volume that occurs in milling operations is not normally appreciated. The informed operator will usually allow for this variation by setting the feeders to produce a smaller amount of material than the maximum capacity of the process. Thus maximum production, as measured by feeder capacity, is not usually achieved.

b) *Gravimetric feeders* fall into two general types, weighing belts and weighing hoppers. The weighing belt or steel apron type gives a good uniformity of discharge, while the weighing hopper yields a discharge by weighted batches.

3)—*Range of feed regulation* is the ability to change accurately the rate of feed during operation and this range does not have to be large under ordinary circumstances.

C—Costs

1)—*Capital Investment*
2)—*Maintenance*
3)—*Power Requirements*

An economic balance between these cost factors must be arrived at, within the physical limitations imposed by the character of the ore and the operating conditions.

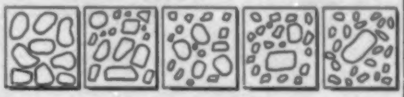
To Determine Throat Opening:

1. Select picture of type of material to be fed.

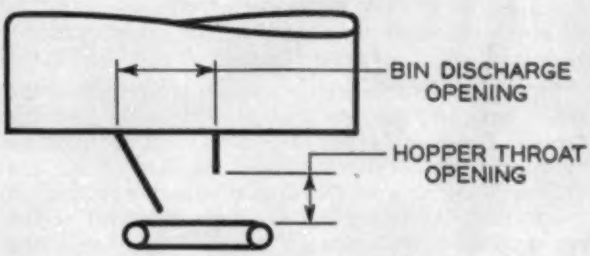
2. Multiply throat opening factor by largest dimension of biggest piece to determine minimum hopper throat opening to prevent bridging.

THROAT OPENING SELECTION CHART

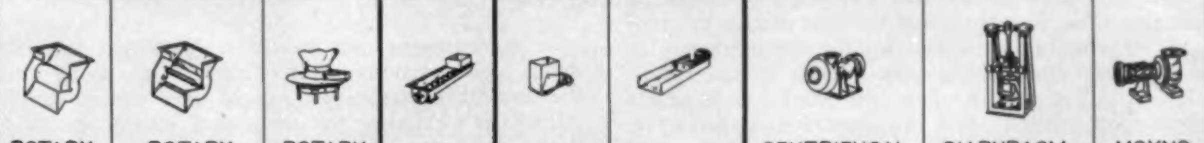
ORE TYPE →



THROAT OPENING FACTOR	$3\frac{1}{2}$	3	$2\frac{1}{2}$	2	$1\frac{1}{2}$
PCT LARGER SIZE PIECES	95–100	60–80	40–60	20–30	5–10
OTHER CONDITIONS	SCREENED	MINE RUN	MINE RUN		

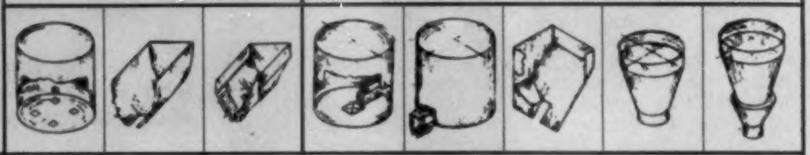


(Shading: Not used — Stipple: Not good)



ROTARY			SCREW	MISCELLANEOUS		PUMPS		
ROTARY DRUM	ROTARY OPEN TYPE	ROTARY TABLE	SCREW	CHOP	PLUNGER	CENTRIFUGAL SAND PUMP	DIAPHRAGM PUMP	MOYNO PUMP
√	√	√						
√	√	√		√	√			
√	√	√	√	√	√			
√	√	√	√	√	√			
			√					
						√	√	√
		√						
		√						
√	√	√	√	√	√			
LOW	AVG	LOW	HIGH	GOOD	HIGH	AVG	LOW	AVG
GOOD	GOOD	GOOD	AVG	AVG	AVG	AVG	AVG	GOOD
AVG	AVG	AVG	AVG	POOR	AVG	POOR	*GOOD*	GOOD
AVG	AVG	GOOD	AVG	POOR	AVG	POOR	GOOD	GOOD
LOW	LOW	AVG	AVG	LOW	AVG.	AVG	AVG	HIGH
AVG	AVG	LOW	HIGH	LOW	HIGH	HIGH	LOW	HIGH
LOW	LOW	AVG	AVG	LOW	HIGH	HIGH	LOW	AVG

Ore characteristics are listed at left. Check means satisfactory application; stipple indicates unsatisfactory application. All bins are bottom opening, except fifth, which is side opening.



BIN DISCHARGE OPENINGS SELECTION CHART								
	MULTIPLE OPENINGS			SINGLE OPENINGS				
BINS →								
FREE FLOWING	√	√	√	√	√	√	√	√
TENDENCY TO STICK	√	√	√	√		√	√	√
STICKY	√		√				√	
FLOODING		√						
SEGREGATION	√						√	
BIN-SELF CLEANING			√			√	√	√

A few tricks of the trade: Ideas that have been used with more or less success to keep ore moving in a bin are listed below:

1. **Bin Wall Movement.** 2. **High Pressure Air.**
3. **Mechanical Agitation.** 4. **Manual Poking.**
5. **Pulsating Pneumatic Panels.**

Feeder Hoppers

The function of a feeder hopper is to enclose the space between the bin discharge opening and the feeder. For many types of feeders, this ore passage or opening is constricted from the size of the bin discharge opening to the area of the ore resting on the feeder. At this point the path of travel of the ore makes a 90° change in direction, and the ore passage is then further constricted by a hopper throat opening whose purpose is to meter and shape the ribbon of ore on the discharge end of the feeder.

For all but free flowing ores, these constrictions can be a source of trouble. For slightly sticky or sluggish ores, the design of the bin discharge constriction should be guided by the same principles as discussed for bin bottoms, having special reference to valley angles. For very sticky ores, there is no good solution, and the constriction should be eliminated by making the feeder a part of the bin.

The selection chart, p. 133, to determine hopper throat opening dimensions to prevent bridging is based on maximum particle size. Use of an adjustable gate at this point is recommended to adjust the feeder capacity to requirements and consideration should be given to the opening shape, as this affects the bedding or ribbon of ore on the discharge end of the feeder. The throat opening factors of 1½ and 2 should be used with caution, as in many cases it is advisable to screen out the oversize, particularly to arrive at an economical size feeder for low rates.

Some design pointers: For certain types of feeders, the weight of the column of ore resting on the feeder has a marked tendency to deaden the feeding action. Where this is true, the hopper should be designed to take the weight, rather than the feeder. Hopper sides and skirt boards should be tapered outward to give relief to the material as it moves forward on the feeder. For corrosive ores it is advisable to consider a type of liner plate that will resist corrosion and *polish up*, thus lowering resistance to sliding. All hoppers should include a bin close-off gate so that repairs can be made to the feeders without having to unload the total bin. These close-off gates need not be elaborate, as they will seldom be used.

Feeder Drives

An important part of feeder operation is the close control of the feeder tonnage to compensate for the changing characteristics of the ore. Such control can be obtained either by adjusting the throat openings or by varying the speed of the feeder. It probably can be said that *all* feeder drives should include a means of making speed changes.

For those feeder applications where a vari-speed drive is required to accomplish feeder control, there are quite a number of drive arrangements from which to choose:

1. V-belt drive with vari-pitch sheaves. This is one of the simplest mechanically, though not adequate for all installations. Constituent parts can be easily assembled by the operator and packaged units are readily available. Similar characteristics are true for a **chain drive.**

2. Ratchet and pawl drive. This type provides a large reduction ratio along with variable speed at an economical cost.

3. A dc-motor and rheostat. This old reliable drive has relatively high efficiency when used in the middle of its speed range. But unless direct current is available, the additional investment of a motor-generator set or rectifier is required.

4. An ac motor with fluid coupling. This is available as a unit or can be assembled by the operator.

5. A modified form of a slip drive with magnetic field replacing a fluid coupling. Small amounts of direct current from rectifier tubes supply the excitation current in the magnetic field, and by varying this excitation a speed variation is obtained. As with all fluid clutches, efficiencies fall off rapidly in the lower speed ranges.

6. Modified wound rotor motor in ac vari-speed machine. The primary and secondary windings are reversed, and speed variation is obtained by shifting the brushes on a commutating winding, thus varying the voltage in the secondary stator winding. Here also, efficiencies drop in the lower speed ranges.

Variable speed prime movers can be classified as constant horsepower or constant torque machines. The V-belt, the dc-motor drive, and ratchet and pawl method are examples of constant horsepower, while the others mentioned above are constant torque. For the usual feeder application, where only small speed variations are required to smooth out the daily changes in ore character, either type is suitable. But for the occasional instance where speed variations are large, the constant torque machine is preferred.

Expanded Perlite Shows Steady Production Growth

by Oliver S. North

RESERVES of perlite rock in the western section of the United States are immense. A geological report prepared for the Union Pacific RR showed proved tonnage of over 400 million tons in southern Nevada alone. Yet the perlite industry is not large, and most of the mines are relatively small.

There are indications that perlite processing, thus far largely in the hands of relatively small companies, is becoming concentrated in a few large companies. For economic reasons the bulk of perlite sold in crude form has come from three or four producers, although many individual operators have sold small quantities to limited lists of purchasers.

Perlite is one of ultra-lightweight aggregates. Owing to its outstanding qualities as an aggregate, variety of uses, and ultimate economy of use, it has enjoyed considerable popularity in this country in recent years. Marked foreign interest also has been evinced in this material.

Huge quantities of perlite have been identified in New Mexico, Utah, Arizona, California, Colorado, and Oregon. No deposits of perlite rock have been found east of the Rocky Mountains, where geologic conditions favorable to its formation are not known to exist.

Growth of an Industry

Virtually unknown on a commercial basis before World War II, production of expanded perlite in 1953 was 43 times greater than in 1946, reaching 174,461 short tons valued at $8,894,735. This remarkable growth of an industry is shown in Table I.

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PERLITE BENEFICIATION

An article on perlite beneficiation: *Processing Perlite—The Technologic Problem*, by Robert H. Weber, appears in the transactions section of this issue, page 174.

Twenty-one firms and individuals in seven western states reported output of crude perlite in 1953. Of the 198,751 tons of crude used during the year, 80 pct was produced in New Mexico and Nevada, 16 pct in California and Colorado, and 4 pct in Arizona, Oregon, and Utah.

Production of expanded perlite in 1953 was reported from 79 plants in 30 states. Of these, 15 were in California, 6 in Texas, 5 each in Illinois, New York, and Pennsylvania.

The mill value of crude perlite (crushed and sized) sold by producers averaged $7.59 per ton in 1953 compared to $6.46 in 1952, while the average value of expanded perlite in bags at the plant was $50.98 per ton in 1953 compared to $51.74 in 1951.

Petrology

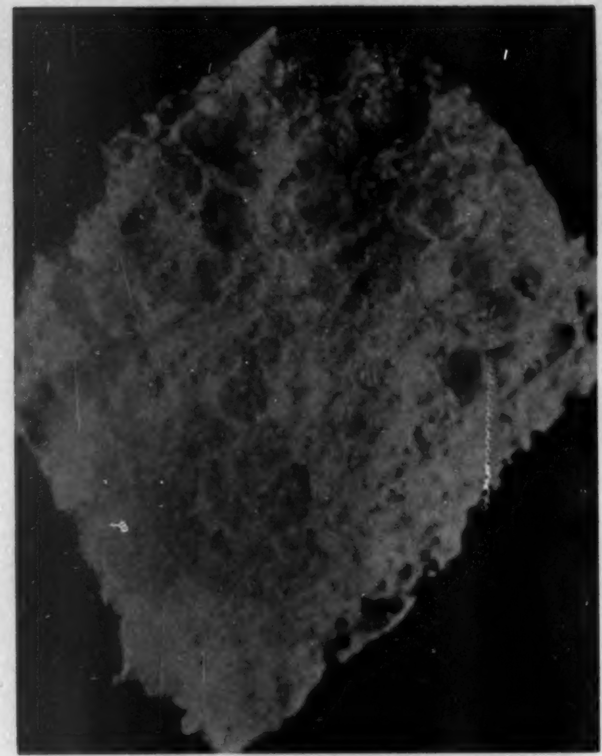
Almost any volcanic glass will show some expansion when subjected to high heat but usually the expansion for pumice, scoria, pitchstone, and the like is not too marked, being 10 to 200 pct. However, most perlite will expand 400 to 2000 pct—a fact that should be accounted for in any commercially justifiable definition.

A Definition: Perlite, as that term now is used commercially, is a rock of volcanic origin, containing chemically bound water and perhaps gases and other liquids, which when suddenly heated to a

Table I. U. S. Crude and Expanded Perlite Production, 1948 to 1953*

Year	Crude perlite: Produced, Tons**	Crude perlite: Sold, Tons	Crude perlite: Sold, Value, $	Used at own plant to make expanded material: Tons	Used at own plant to make expanded material: Value, $	Expanded perlite: Produced, Tons	Expanded perlite: Sold, Tons	Expanded perlite: Sold, Value, $
1948	22,200	4,400	29,000	17,700	105,000	21,200	18,600	742,000
1949	71,500	27,300	193,000	43,800	317,000	58,100	52,200	2,385,000
1950	110,694	59,802	411,205	41,734	237,957	88,892	86,962	4,741,383
1951	182,260	110,119	663,981	43,383	194,118	134,479	133,175	7,243,298
1952	190,442	135,070	873,054	29,775	129,866	155,955	154,563	7,997,731
1953	213,532	141,282	1,072,065	57,469	367,593	175,234	174,461	8,894,735

* Source: USBM, *Mineral Market Report MMS 2329.*
** All figures in short tons.



Micrograph shows glass sealed air cells that give perlite its properties of lightness with strength and fire resistance.

suitable temperature in its softening range undergoes great expansion owing to the volatilization, within the softened mass, of the gases and/or liquids; such expansion varies from 400 to 2000 pct or more, depending on the inherent nature of the rock and controlled furnacing factors.

As pointed out by C. R. King, "Perlite is a rock, not a mineral, and therefore is variable in chemical composition within a wide range. Composition of separate deposits differs, and there is wide variation even within the same deposit."[1] The chemical composition of a given perlite is very nearly the same, except for the combined-water content, as that of the volcanic rock to which it is genetically related.

The quantity of combined water in a perlite is important, although King, Todd, and Kelley state: "The measurements show that the bulk of the water in perlite is relatively loosely held, but that as dehydration proceeds the residual water is held progressively more firmly."[2] They conclude that the water above 1.2 pct is "loosely held." Whether or not the chemically combined water above that percentage contributes to the expansion is uncertain; perhaps it is evaporated before any considerable expansion occurs. It may be that only a small part of the plus-water is active in causing expansion.

Theories on the mode of origin of perlite differ to some extent; in fact, no one may be correct for all deposits. The formation of perlite as a surface extrusion has been questioned, and most geologists who have studied the matter lean either to near-surface intrusion, which would permit fast cooling under moderate pressure, or to hydrothermal alteration of other materials, such as rhyolite or pumice.

In hand specimens perlite is difficult to identify. It has wide color and textural range; the perlitic structure may not be visible; its specific gravity, although somewhat lower than for most volcanic rocks, is not distinctive; its luster, usually but not invariably *pearly*, approximates that of many siliceous rocks; and its rock associations, formerly considered exclusively rhyolitic, are now believed widely variable. Unless from an area previously known to contain perlites, where its general appearance may be well known, it can be identified with certainty only by testing for degree of expansion.

Mining and Processing

Virtually all perlite is mined by open pit methods. Mining is simple, with the exact details dependent on specific locality. Required are a means of moving and disposing of overburden, equipment to break rock when hardness requires it, and a means of loading and moving the perlite to the mill.

Milling is also relatively simple. It consists of crushing the perlite rock and separating the product on the basis of desired size specifications. Two processes have been used, one wet and the other dry. The wet method minimizes the dust hazard but necessitates a more expensive drying procedure and requires large quantities of water, an item often in short supply in the vicinity of perlite deposits. As a result, virtually every perlite mill now uses the dry process.

After primary crushing in a jaw or impact crusher the material is sent to a secondary unit, rod mill, ball mill, or rolls. Surface moisture is removed in a flash-drier, and the material then screened. Screen oversize is further ground and screened in closed circuit. Ground rock is separated into the desired gradings by vibrating screens. Extreme fines are either sent to waste or stored with an eye to possible future demand for that fraction. Major producers have installed belt conveyors for transferring the crushed rock, and blending the material is regular practice. The larger firms that sell furnace-grade crude perlite to other expanders use automatic samplers.

Several types of furnaces in which perlite may be expanded have been developed, but two basic types are most widely employed—the horizontal (rotary and stationary) and the vertical (stationary.) Each has many variations of detail, and no one furnace can be considered best for every type of crude perlite used and end product desired. It is generally considered that horizontal furnaces handle a wider variety of perlites and take a coarser-grained feed.

Table II. Expanded Perlite Production and Sales in 1953[1]

State	1953			
	Produced	Sold		
	Tons	Tons	Value, $	Avg $ per ton
California	35,403	35,342	1,601,988	45.33
Illinois	11,127	11,127	712,238	64.01
Ohio	10,344	10,015	675,207	67.42
Pennsylvania	13,158	13,109	810,965	61.86
Other Western States[2]	49,680	49,253	2,194,613	44.56
Other Eastern States[3]	55,522	55,615	2,899,724	52.14
Total	175,234	174,461	8,894,735	50.98

[1] Source: USBM *Mineral Market Report MMS 2329.*

[2] Includes Arizona, Arkansas, Colorado, Iowa, Kansas, Louisiana, Minnesota, Missouri, Nebraska, Nevada, New Mexico, Oklahoma, Oregon, and Utah.

[3] Includes Florida, Indiana, Maryland, Massachusetts, Michigan, New Jersey, New York, North Carolina, Tennessee, Texas (1952 only), Virginia, and Wisconsin.

LEFT: Hill of perlite near Colorado Springs, Colo., was formed from cooling lava. Shallow or surface pit is typical of perlite operations. RIGHT: Storage bins and crushing and sizing facilities as at this mine in the West are usually extent of preparation plant located at mine site.

Ordinarily they are used for longer periods of retention of rock in the furnace, producing *intermediate* to *dead* perlites. Vertical furnaces have short retention times and few moving parts. Most furnaces deliver the expanded perlite to a cyclone collecting system in which the product is sized. In some plants the fines are collected in a wet dust-collector and the slurry sent to waste. In most cases gas is used, although some furnaces are oil-fired.

In any furnace the finished product is influenced by size and rate of feed, temperature, and retention time, as well as by physical and chemical nature of the crude rock.[3]

Perlite Utilization

About 80 pct of U. S. output of expanded perlite is used as an aggregate in job or premixed gypsum plaster. Advantages claimed for perlite plaster include light weight, good acoustical and thermal insulating properties, fireproofing, resiliency, nailability and sawability, ease and rapidity of application, good bonding properties, etc. Perlite-gypsum plasters are of three major types: regular hardwall plasters, fireproofing plasters, and acoustic plasters.

Another 10 pct of perlite production becomes aggregate in concrete. Most important uses at present are roof decks, prefabricated panels and sections, and floors. In Pittsburgh four large office buildings were built with exterior wall slabs of aluminum or steel sheets backed up with perlite concrete.

Numerous uses for perlite have been developed, including the following: 1) As an additive to drilling muds (about 4 pct of production). 2) As loose-fill insulation in concrete block walls and between wall studs. 3) As an insulator for steam pipes, either as loose-fill in jackets, or as a layer of plaster. 4) To insulate refrigerators, etc. 5) As a loose-fill medium for imbedding hot steel ingots during shipment. 6) As a substitute for ordinary foundry sand to surround a pouring riser. In this application the insulating properties of the perlite reduce the rate of cooling of the riser. 7) In glazed building tile, sandwich boards, metal surface plaster, and insulating brick. 8) As a filler and extender in rubber, soaps, paints, plastics, resins, etc. 9) As a filter aid, which now consumes approximately 2.5 pct of production.

It was soon discovered that economic advantage lay in shipping the crude rock to areas of ultimate consumption and there expanding it, rather than in furnacing it at the mine and shipping the expanded material. Freight rates and other practical considerations regarding availability of fuel, labor, etc., have resulted in almost universal adherence to the practice of processing at the center of consumption.

But, it was found advantageous to crush at the mine and to ship furnace-grade material rather than lump rock. The principal reason for this is the widely scattered locations of the processing plants, each of which uses a relatively small tonnage.

Marketing Perlite

Transportation costs are a large part of the delivered cost of crude rock. Even higher freight charges for the bulkier expanded product tend to limit its area of distribution to a 50 to 200-mile radius around a plant. Companies try to locate new furnace facilities in the immediate vicinity of concentrated construction activity.

The prices for crude perlite rock, f.o.b. mill, ground to customer's specifications, usually range between $6 and $14 per ton. Plaster and concrete aggregates, in 3 or 4-cu ft paper bags, sell for from 15¢ to 40¢ per cu ft.

Although there have been inquiries into the possibility of importing crude perlite into the East, little if any now is imported, and the quantity of processed perlite exported also is small. One firm reported shipping expanded perlite to Venezuela for use in oil wells, and minor quantities probably leave the U. S. as filler in various products. An increasing quantity of crude perlite is being exported to processors in eastern Canada.

This industry is so new that laboratory research has been at a minimum. The USBM has studied the thermal characteristics of perlite,[2] and the National Bureau of Standards collaborated with the Bureau of Reclamation in a study of the characteristics of many lightweight-aggregate concretes, including perlite concretes. At least one college in Texas has experimented on its use in oil wells and many perlite expanders conduct tests and do experimental work on their own product. The industry cooperates through the Perlite Institute, which releases technical data to its members and to users.

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Scrubbing Solves Sand Flotation Problem

Flotation permits Del Monte to produce wide variety of high quality sand products from unusual Pacific Coast sand deposit.

by William E. Messner

CALIFORNIA'S Monterey Peninsula, noted for its beaches, resort hotels, and beautiful homes, also harbors a pioneering beneficiation operation. Here for more than 35 years the Del Monte Properties Co. has provided sand products from large dunes, some over 300 ft high, located in the Del Monte Forest.

Raw material—the dunes derived by wave action from Santa Lucia granite of Jurassic age—has unusually consistent chemical and physical composition. Coarse crystalline feldspar is the principal constituent, followed in abundance by quartz, with some biotite, ilmenite, garnet, zircon, and monazite.

Grain Size of Del Monte Raw Sand

Screen Size, Mesh	Pct
+20	trace
40	3.5
60	62.3
80	31.0
100	2.2
−100	1.0

Raw sand ranges from 0.12 to 0.18 pct Fe_2O_3 and averages 11.5 pct Al_2O_3, with a screen analysis of all −20 mesh, and 1 to 2 pct +100 mesh.

Research work on flotation methods has been carried out since 1938 by groups interested in the chemical content of the sand for West Coast glass and ceramic uses. In 1942 the Owens-Illinois Glass Co. built a pilot plant at Del Monte to separate feldspar and quartz. Although flotation was abandoned in favor of a magnetic separation plant, the data was later used by the Del Monte Properties Co.

In 1948 American Cyanamid Co. became interested in applying its 800-series reagents for iron removal. Encouraging laboratory work was followed up with a 2-tph pilot plant in 1949, built by Del Monte in cooperation with American Cyanamid. Pilot plant operation showed unsatisfactory sand recovery after iron removal until the introduction of scrubbing at high solids to remove surface stains and clean the sand grains prior to flotation. This vital step raised recoveries from as low as 60 pct to 90 or 95 pct.

W. E. MESSNER, now deceased, was formerly Plant Manager of the Sand Div., Del Monte Properties Co., Pebble Beach, Calif.

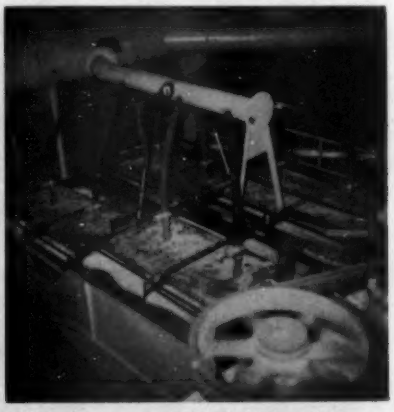
Key is Attrition Scrubbing

A modification of the attrition machine developed by Owens-Illinois at its Corona, Calif., sand plant was built for Del Monte by the Western Machinery Co. Primary function of the unit is scouring and scrubbing without particle size reduction. All parts are rubber-lined to resist wear, and each cell has individual V-belt drive.

Designed to handle 50 tph at 75 pct solids, the attrition plant started operation in 1950. Flowsheet called for ⅜-in. wire cloth on a 4x12-ft Allis-Chalmers vibrating screen, an 80-ton surge bin, three Stephens-Adamson reciprocating feeders, and three attrition units with total plant connected load of 263 hp. Each attrition unit was composed of six Wemco scrubber cells and a 30-hp, 4-in. Wemco sand pump.

Flotation plant production began in January 1952. Two section operation provides iron removal and rejection ahead of a quartz-feldspar separation. Scrubbed pulp is diluted to 25 pct solids and screened at 20 mesh on a 4x5-ft Ty-Rock screen. Oversize goes to waste and −20 mesh is cleaned in a 48-in. Wemco spiral classifier to remove organic and inorganic slimes as thoroughly as possible prior to flotation. Classifier sands at 80 pct solids go to a 12x8-ft rotary drum where they are conditioned for the iron removal step.

Reagents for iron removal added at the conditioner are: fuel oil, sulphuric acid, American Cy-



LEFT: Wemco attrition machines provide first step in preparation. CENTER: Drum conditioner is operated at high solids. Screen and dewatering screw show in background. RIGHT: These Steffenson flotation cells are used for quartz-feldspar separation.

Loading sand from Cypress Point dunes to feed the Del Monte flotation plant.

anamid 800 series reagents, and pine oil. The 2.5 pH is critical at this point.

Iron Removal

After dilution to 25 pct solids, the conditioned pulp is pumped to the twelve No. 60 Steffenson air operated cells in the iron rougher circuit. Highly abrasive and corrosive pulp conditions are met by wood and stainless steel cell construction. Iron rougher concentrate consisting of biotite mica and heavy minerals, is subsequently reduced in weight in five No. 20 Steffenson cells where the iron cleaner concentrate or overflow is rejected and the tailing or underflow is returned to the head of the circuit. Plant capacity is approximately 40 tph of sand averaging 11.5 pct Al_2O_3 and only 0.06 to 0.065 pct Fe_2O_3. This cleaned sand is the iron rougher tailing and may be dewatered and dried at this point for shipment, or pumped to the quartz-feldspar separation circuit.

Quartz-Feldspar Separation

After iron removal the sand is pumped to a 36-in. Wemco spiral classifier for removal of previous flotation reagents. Classifier sand discharge at 80 pct solids goes to three 3x6-ft vertical conditioners, is diluted to 50 pct solids with fresh water, and conditioned with hydrofluoric acid, Armac T amine, kerosene, and pine oil. After dilution to 25 pct solids the conditioned pulp goes to eight No. 20 Steffenson cells for flotation at 2.8 pH. Hydrofluoric acid acts both as a control for pH, which is critical, and as depressant for quartz. Concentrate or overflow from this step is feldspar of 19.5 to 20.0 pct Al_2O_3 and 0.10 pct Fe_2O_3, and the tailing or underflow is quartz containing 1.0 pct Al_2O_3, 0.025 pct Fe_2O_3, and about 98 to 99 pct SiO_2.

Sand transfer between circuits is taken care of by six 4-in. and three 3-in. Wemco sand pumps. Corrosion and abrasion are the primary maintenance problems, and rubber compounds are under continuing test in an effort to lessen these problems. Flotation air requirements of 10,000 cfm at 5 psi are met by three 18x30 Sutor blowers powered by three 125-hp U. S. motors. Addition of eight different reagents at ten stages in the flowsheet is carried out with accurate metering equipment.

After drainage to approximately 5 pct moisture content all products go to two gas-fired 5x16-ft Standard Steel Co. dryers. Three Tyler Hummer screens size the dried product which then goes to storage for feed to sacking machines, bulk carloading, or the grinding operation.

Ground Products

About a year ago Del Monte doubled grinding capacity by adding an independent grinding section with a 10x8-ft Hardinge pebble mill. The older grinding section has two 8x5-ft Hardinge pebble mills. The new mill has a constant weight feeder controlled by Hardinge Electric Ear, and is of venturi type, with the discharge flowing into the airstream of a 40-hp Clarage fan. Mill oversize is separated in a No. 90 Hardinge classifier and returned to the mill. A 10-ft diam product collector delivers undersize to three 60-ton conical steel storage bins. Four St. Regis sacking machines are used in the grinding plant, and product may also be bulk shipped.

Product variety results from plant flexibility, and in addition to the iron-floated material, and the quartz and feldspar from the second or separation section, blended products are available. For example, Del Monte "66" containing 8.0 to 8.5 pct Al_2O_3 and Del Monte "77" with 6.5 to 7.0 pct Al_2O_3 have been successfully used in the glass container industry without further sand additions. These products result from taking iron-floated material plus quartz from the second section and blending in pumps before dewatering. To complete the range of products, the various materials are available either as sized, or in ground form.

Discovery Thinking In Ore-Search

by Thomas W. Mitcham

NUMEROUS ore deposits remain to be discovered. Many exposures of favorable host rock bodies within mineral provinces have not been adequately scrutinized geologically—the ore potential in buried portions of such rock bodies is enormous. Inasmuch as the more obvious deposits have been found, exploration has become more scientific and more expensive. Yet discovery continues to be very rewarding if exploration is intelligently approached.

Although a number of exceptions can be cited, systematic geological exploration for ore on a significant scale has developed during the last two decades. So rapid has been the development of systematic exploration that a comparison of modern programs with those as recent as 1940, would suggest that many large mining companies and various Government agencies were hardly in the exploration business at that time.

What is modern ore exploration? What major factors affect its success—economic discovery?

Definition of Exploration

Efficiency and proper organization of any effort is dependent on clear understanding of its meaning and purpose. Such understanding is aided by concise definition of that effort. Modern exploration for ore is a program involving three essential steps which must be in the sequence listed below.

1. Study of the occurrence characteristics of orebodies of various types—considering known deposits as they occur within a district or province—comparing these with other deposits in other areas.
2. Rational application of knowledge gained from the above study to predict favorable loci for orebodies.
3. Physical testing of these loci for the presence of ore when testing appears economic.

While the primary objective is economic discovery, the above steps can be considered as secondary objectives. Study of literature and aerial photographs, geological mapping, mine mapping, genetic considerations, sampling, laboratory studies, geophysical studies, driving opening into the ground, property acquisition, etc., are *techniques*—not exploration objectives. No one technique or combination of techniques is exploration; exploration is the intelligent use and integration of these techniques.

A number of exploration programs appear to reflect lack of basic understanding. Three errors are common in such programs: 1) techniques are confused with objectives; 2) the three basic steps are out of proper sequence; 3) one or more of the basic steps is omitted.

Exploration vs Measurement: Older methods of exploration stressed a careful and tedious assessment of measurable ore; consideration of potential ore, if mentioned, received scant treatment. Generally, old examination reports included brief discussion of geology as a subject of incidental interest. Only rarely was the geology brought to a conclusion and used to evaluate the property. Unfortunately, some modern examination reports closely resemble these old reports and completely lack vision.

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Property owners often ask prices today which exceed even the gross value of measurable ore. It is very rare that a significant amount of ore can be measured. Thus, only an approximation of the value of measurable ore is usually desirable. Undue stress on measurable ore is somewhat like assessing a man's wealth by counting the money in his pocketbook.

Exploration vs Geology: Exploration involves the focusing of geological knowledge. The simple performance of a good geological job is not exploration.

Comprehensive and thorough geological knowledge of an area is an excellent point of departure for exploration within it. Every new bit of geological knowledge concerning an area may affect the approach to exploration there. The U. S. Geological Survey and many State agencies have performed a commendable service to exploration, particularly in the areal extension of geologic knowledge. Few exploration geologists will enter a new area without reference to standard publications on it, if such basic studies are available. The universities, in particular, continue to lead the advancing thought frontiers of geologic knowledge.

While the exploration geologist should be informed on the advance of geologic knowledge in general, he must remember that his job is search for ore and not performance of geologic studies. One cannot equate exploration and geologic study. The definition of exploration as given above can aid the geologist in defining which areas of geologic knowledge are applicable to ore-search.

An obstetrician is a medical specialist who delivers babies; he does not simply make biological studies. An exploration geologist is a specialist who delivers orebodies; he does not simply make geological studies.

Time and Space

Exploration can be considered as the search for and the application of significant spatial relationships between geologic features and orebodies. In a limited sense, geologic features, suggesting the nearby presence of ore, were the guides of the sourdough prospector. The geologist, through increased power of observation assisted by deductive reasoning, has an increased efficiency which greatly reduces the risk of exploration.

Time relationships apply to exploration when they can be resolved to spatial relationships. Thus while paragenetic studies are of value to the science of ore deposits, they seldom can be applied to ore-search. Genesis of ore, which involves time and space, is a desirable consideration. However, it is not necessary to apply the knowledge of it to exploration. On the other hand, knowledge of genesis can bring new spatial relationships into focus. A statement by Locke illustrates the importance of stress on spatial relationships ". . . no amount of reasoning that ore ought to occur at a given intersection will take the place of the fact that, in that district, or in that mine, it habitually does so."

Generally, an exploration report is lacking of conclusion unless it describes or logically speculates on the application of significant spatial relationships.

In mining exploration, rewards are great for the courageous. Virtually worthless ground can become priceless. High risk is forever present, but good geological exploration will substantially reduce it.

There is no place in exploration for fearful men. A timid motorist, arriving at a busy street intersection, calculated the probability of collision—considering his possible acceleration and chances for motor failure; the speed, frequency, and braking power of passing vehicles. Finally convinced that the risk was too great, he did not cross. Another man at the same point considered the desirability of crossing, quickly balanced risk and desirability, and crossed safely. Both lived to old age, but the former crossed no streets.

Survival of the Courageous

The best geological reasoning should be demanded in support of expenditures on physical exploration. When such reasoning logically develops a fair probability of hitting an economic target, physical exploration should be undertaken without hesitation.

A method of detecting flying ducks at night had been developed. As birds flew by, pips were picked up on a radarscope. The pips were of equal size and intensity for all types of birds. However, study of the radarscope to determine speed and flight pattern would enable the observer to determine whether or not the objects were ducks to a fair degree of probability.

A night duck-hunting contest was held. Contestants were to pay $1,000 per shell fired; the prize money was $100,000 per duck shot down. Contestants were to form groups of three men each. A radar set was available to each group for a fee of $5,000 for the night.

Many groups participated in the one-night contest. The groups used four different methods of approach in their attempts to win prize money. The methods of four example groups are described below:

Group 1—These men observed 100 pips, and after quick calculations, intelligent reasoning, and discussion on each pip, they determined that 20 were probably ducks. They fired 20 shells, bagging 10 ducks and 10 sparrows. Shell costs plus radar rental were $25,000; their prize money was $1 million.

Group 2—This group also observed 100 pips and determined, in a most intelligent manner, that 15 were probably ducks. However, they reasoned that each shot was, nevertheless, a risk. They fired no shells and lost only the radar set rental, $5000.

Group 3—These men determined that the tenth pip on the screen was a probable duck. They fired and bagged a sparrow. Frightened, they turned in their radar set and went home, losing only $6000.

Group 4—These men reasoned correctly that a radar set could not bag ducks, no matter how intelligently it was used. After all, one could shoot five shells for the rental cost of a radar set. They shot 200 shells and bagged only two sparrows. They saved the $5000 radar rental but lost $200,000. Particularly disheartening was the fact that the majority of the contestants followed this approach. Contest officials explained that in a similar contest three years ago, one group had bagged 15 ducks by this approach. Their success had influenced hundreds of other groups to lose heavily.

In conclusion on night duck hunting, the following points are made:

1. If hunters lack the courage to fire, at least on occasion, no amount of intelligent deliberation will bag ducks.
2. To misuse or ignore modern scientific knowledge on night hunting is a foolish way to hunt.

Reduction of Risk: Risk can be reduced by a systematic exploration program guided by good geologic thinking. Geophysical methods are additional geologic techniques which can sometimes aid in the reduction of risk, but it should be stressed that instruments and the mathematical results derived from their readings are in no way substitutes for geologic thinking.

Vendors of mineral rights can aid in reducing risk if they can be convinced that chances for payment are greater when adequate time is allowed to properly test ground before the first payment is required. Sellers should, of course, be protected by adequate work requirements, assuring that the property will receive adequate testing. When one buys an automobile, he is not expected to pay for looking under the hood to see if the motor is installed. Likewise, one should not require payment prior to examination and preliminary testing of mineral land.

In the last two decades, ore-search has become more expensive, and more urgent. Tax laws should be further revised to encourage exploration. Increased exploration activity will increase national reserves of mineral raw materials. The Revenue Act of 1951 was a step in the right direction, but more liberal allowances for exploration deductions are needed.

The Program

Exploration for ore should be geologically guided and directed. This does not necessarily mean that the exploration man should have one or more degrees in geology. It does mean that he must have keen geologic knowledge. A mining engineer who is a good geologist is usually an excellent exploration man.

The ideal exploration program should demand that a certain percentage of appropriated exploration funds be spent each year for preliminary physical testing of ground. Expenditures on testing should at least equal expenditures on geologic investigation. Such a plan sets goals for geologic investigations.

In conclusion, an exploration program will be successful if the exploration rules discussed in this paper are observed. These can be summarized as follows:

1. Place primary emphasis on geologic thought which is directed to the proper conclusion—prediction of favorable loci for ore.
2. Emphasize the definition and thus the objectives of exploration for ore.
3. Courageously apply intelligent reasoning and adequate funds to financing of the program.

It is most important that the emphasis of the exploration program be on thinking. Hundreds of thousands of tons of structural steel and hundreds of thousands of man-hours were expended on the construction of a bridge. Upon completion of the project, the bridge collapsed. Because of lack of thinking, the enormous physical effort was a complete loss. Thinking on exploration is more abstract, and its emphasis in exploration programs is even more important. Feet of drilling, tunneling, or sinking is never a measure of performance of an exploration project. One geologic conclusion or concept can make the difference between exceptional success and total loss.

Are Engineers Prepared For Executive Responsibilities?

by A. C. Dorenfeld

The cry is heard almost daily, "We need more engineers!" But is this need being filled with professional men or with quasi-educated technicians?

IN most mineral enterprises, what is the progress, and shift in responsibilities, as the young engineer advances in the corporation? You are all familiar with the normal pattern—in mine production from level boss, to shift boss, to superintendent to manager; in milling from test engineer, to shift boss, to foreman, to superintendent and then manager; in engineering from surveying on to Chief Engineer; and all then aspire to the corporation presidency. Note the shift—from purely technical jobs, say in the first five to ten years in the profession, to administrative positions for 30 to 40 years.

What is the average preparation given to mineral engineering students, who can look forward to about 30 years of administrative work? From the catalogs of 21 institutions, offering 36 curricula of four years length, in mining and/or metallurgy, requiring the equivalent of about 150 to 160 semester-hours for graduation, the following results were obtained: Average number of semester hours of economics, 3 hr. Range of requirements, 0 to 8 hr. Thirteen curricula do not require any economics. Accounting and business law, average: 1 semester hr, 26 curricula do not require business law or accounting.

It is well recognized that the mineral industry, while an important part of the general economic pattern, has peculiar economic problems of its own, such as depletion problems, international exchange, international control of minerals, etc. Yet the average time devoted to this subject is 1 semester hr, with *23 curricula not requiring such studies.*

The implications are clear—a great majority of our engineering training is most useful for about 25 pct of professional life, and then becomes of decreasing importance—while an insignificant amount of training in school—largely nonengineering subjects, is most useful in later professional life.

What are the reactions of engineering graduates and of managements to these facts? Are they aware of these facts? In a survey of 200 companies and 1300 engineers, excerpts of which were published in the January 1954, issues of *Chemical Engineering* and MINING ENGINEERING (Executive Research Surveys 1 & 2, National Society of Professional Engineers), the following sentences are quoted:

> Twenty-eight pct of those who replied (that is engineers) said that they did not feel that their collegiate training prepared them adequately for a career in engineering, and most of those indicated a need for more work in English, the social sciences and business administration.

Management opinion may be summed up as follows:

> We are getting too many young men whose narrow training fits them for nothing better than technician's jobs.
>
> They know the engineering principles thoroughly. They can analyze a problem and come up with the right solution. But they do not know how to put their ideas into words, either orally or on paper. Such a man can be valuable in a subordinate capacity, but he is not going very far, in the long run, and it usually does not take him long to discover his own inadequacy, and to resent it. Frequently, he directs that resentment toward his superiors or his fellow workers, and the first thing you know, we have a morale problem.
>
> It seems to me that we in industry are doing ourselves, as well as the engineering profession as a whole, a disservice in not insisting that the colleges offer well-rounded programs which will turn out not just engineers, but educated engineers.

It is clear, then, that dissatisfaction with our engineering education is rife among our graduates and corporate managements. Is this phenomenon new? It is not. Some eighteen years ago, when the author was looking at college catalogs, endeavoring to choose an engineering school to attend, catalog after catalog, stressed, in words, the need for a well-rounded education—eighteen years later practically nothing has been done about it. A few schools have added a fifth year—and that many more technical subjects!

The conclusion is inescapable—engineering schools are *not* turning out graduates equipped with the knowledge to make them leaders in industry and in society as a whole. In general, we are turning out technicians—men able to do technical work—but not equipped to *direct* the fruit of their labor—in short not able to take their place in management smoothly and effectively.

What can be done about it? That something should be done about it has long been recognized, and some work on this score is being done by the American Society of Engineering Education. The proposals have, so far, been of one cloth—add a few more courses in humanities, taught, frequently, with 100 boys jammed into a lecture hall, and under the handicap of a student attitude: "Let's get it over with." In order to teach engineering subjects adequately, in a four-year course, the student load is heavy. The study of humanities thus suffers.

What is the remedy? One simple solution is to make engineering education thoroughly professional, similar to medicine, requiring men entering engineering school to have three to four years of college training, including languages (English as well as foreign), social sciences, and economics as well as the usual physical sciences and mathematics. That is one way we can turn out *educated* engineers, capable of dealing more effectively with today's complex social and economic problems.

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Miniature Oblique Photogrammetry

by A. N. Winsor

INCREASED use of private aircraft in civil engineering, mining, forestry, and other land-use activities creates an opportunity for amateur aerial photography. Taking a photograph having less than 5° tilt requires precision equipment and skilled personnel, while controlled horizontal photography is primarily a terrestrial method that has passed to the limbo of forgotten things. But the remaining possibility—oblique aerial photography—can be

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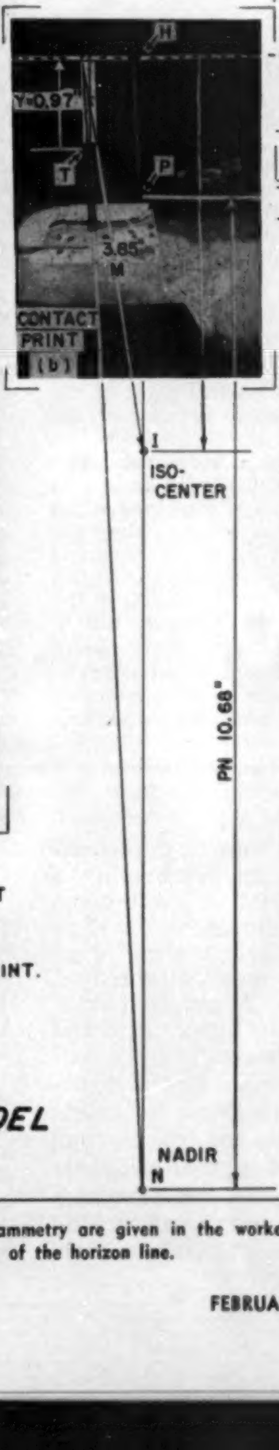

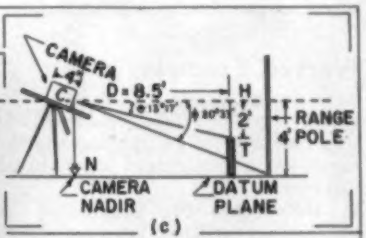

ELEVATION VIEW

$$\text{TAN. } \phi = \frac{HP}{f} = \frac{1.5''}{4''} = 0.375$$

$$\phi = 20^\circ\,33'$$

$$\text{CO-TAN. } \phi = 2.67$$

$$\text{SIN. } \phi = 0.351$$

$$\text{COS. } \phi = 0.936$$

$$HI = \frac{f}{\text{COS } \phi} = \frac{4}{0.936} = 4.27''$$

$$PN = f \text{ CO-TAN } \phi = 4 \times 2.67 = 10.68''$$

$M = 3.85''$ (SCALED)
$Y = 0.97''$ (SCALED)

ANGLE OF DEPRESSION θ FROM HORIZON TO OBJECT "T".

$$\text{TAN. } \theta = \frac{Y \times \text{COS } \phi}{M} = \frac{0.97 \times 0.936}{3.85} = 0.236$$

$$\theta = 13^\circ\,17'$$

VERTICAL DISTANCE, HORIZON TO TOWER "T" IN TRIANGLE CTH

$$\text{TAN } 13^\circ\,17' = \frac{TH}{8.5'}$$

$$TH = 8.5 \times 0.236 = 2 \text{ FT.}$$

HEIGHT OF TOWER $= 4 - 2 = 2$ FT.

2.5 ENLARGEMENT
DISTANCE SCALE, RELATED TO AN ENLARGEMENT REFLECTS THE RATIO OF ENLARGEMENT. THE CALCULATIONS AT THE RIGHT ARE BASED ON MEASUREMENTS SCALED FROM THE CONTACT PRINT.

OBLIQUE VIEW OF SCALE MODEL

Calculations for this scale model demonstration of photogrammetry are given in the worked example on the following page. This view is considered as a high oblique, based on position of the horizon line.

used to advantage by persons who occasionally make maps although not engaged in photogrammetry.

Oblique photographs offer: 1) natural portrayal of terrain, reducing need for skill in photo interpretation; 2) as much as ten times more land area may be covered by a single oblique than in a single vertical photograph from the same altitude; 3) possibility for an amateur to take a usable oblique photograph with an ordinary camera; 4) if the horizon line is included, the camera may be tilted horizontally as well as vertically without loss of photographic control; and 5) reasonably accurate mapping with ordinary drafting tools.

Photographs having a tilt from 5° to 85° are generally considered as obliques of either the high or low variety; here, the terms *high* and *low* do not refer to altitude but to the depression angle of the aerial camera. The high oblique is one with the line of apparent horizon in the photograph. The low oblique shows no horizon trace and is seldom used because such photographs lose, unnecessarily, the control provided by the horizon line.

The accompanying enlarged and contact-size photographs of a scale model demonstrate the photogrammetric principle for determining elevation differences in an oblique photograph. The photograph may be considered as a high oblique because the true horizon line was a predetermined factor of scene and camera station. A camera having a 4-in. focal length was on an inclined plane table with the lens 4 ft above the general level of the area and the horizontal distance between lens and object (2-ft mailing tube) 8.5 ft. In this scale model problem it was desired to calculate the elevation of the top of the mailing-tube tower in reference to the known 4-ft control elevation of the horizon line and the camera lens.

Worked Example:

Calculating the Height of Tower "T"

A representative graphic method of determining the elevation of an object in an oblique photograph is as follows:

a. Determine the depression angle of the camera. The scaled distance between the apparent horizon and the center *P* of a contact photograph determines the apparent depression angle of the camera: Distance *HP* scaled 1.5 in. on the contact print, Fig. b.

$$\text{Tan } \phi = \frac{\text{Distance HP}}{\text{Focal length of camera}} = \frac{1.5}{4} = 0.375$$

$$\theta = 20° \ 33'$$

where *H* is Horizon and *P* is the central point of the photograph lying on a perpendicular bisector of the horizon.

In an aerial oblique photograph a correction must be made for the dip of the horizon. The horizon as seen from a high altitude dips, due to the combined effect of the Earth's curve and light refraction, below the relative altitude of the observer by an approximate angular quantity in minutes equal to the square root of the altitude reading in feet. For example, if the altimeter reading is 5000 ft, the horizon dip is $\sqrt{5000} = 0° \ 70.6'$. This angle is added to the apparent depression angle to obtain the true depression angle, and a trace line, above the apparent horizon, is drawn to indicate the true horizon from which all subsequent measurements are taken. In a miniature oblique photograph the horizon dip is considered to be zero because of the short distances involved.

b. Locate the Nadir *N* and the Isocenter *I*, both lying on the perpendicular bisector of the horizon, see illustration, Fig. b.

$$\text{PN} = \text{f}\ (\text{cotan}\ \phi) \qquad \text{HI} = \text{f}/\cos\phi$$

$$\text{PN} = 10.68 \text{ in.} \qquad \text{HI} = 4.27 \text{ in.}$$

c. Construct rays from the Nadir and Isocenter to the horizon. Draw a ray from *N* to the horizon passing through the object *T*. Draw a ray from *I* to join the previously drawn ray at the horizon.

d. Determine the depression angle, camera to object *T*.

$\text{Tan } \theta = \dfrac{\text{Y} \cos \phi}{\text{M}}$ where Y and *M* are scaled on a contact print as shown, Fig. b.

$$\text{Tan } \theta = \frac{\text{Y} \cos 20° \ 33'}{\text{M}} = \frac{0.97 \times 0.936}{3.85} = 0.236$$

$$\theta = 13° \ 17'$$

e. Determine vertical distance horizon to object and height of object. In the right triangle *CHT*, Fig. c, the expression $D \tan \theta$ is the vertical distance between the horizontal plane, which contains both the camera and horizon, and the object.

$$\text{D} \tan \theta = 8.5 \times 0.236 = 2 \text{ ft.}$$

The height of object equals the camera elevation minus $D \tan \theta$, or $4 - 2 = 2$ ft.

Application

Because of the availability of control, the miniature oblique photograph tends to oversimplify a problem. In practice there would be encountered such uncertainties as the exact altitude of the camera, resulting from an approximate reading of an altimeter, and perhaps an indefinite horizon line. However, an oblique photograph of natural terrain can be adjusted to yield the correct flying height and true horizon, if it contains the images of three well distributed control points. There are several methods of adjusting an oblique photograph for determining the altitude of the camera and true horizon. Most of these methods are based on trial-and-error whereby the correct altitude is that which satisfies three control points. After the correct camera altitude has been ascertained the trace of the true horizon can be drawn and the depression angle of the camera calculated. From these data the nadir and isocenter points of the photograph are located.

A project map can be constructed by plotting the position of objects separately, using the perpendicular bisector of the horizon as a direction line and the nadir point as the pivot for directional rays to objects. Horizontal angles, measured by protractor, at the isocenter of a photograph between the perpendicular bisector and images of objects are transferred as true map angles at the nadir between the direction line and objects. If the terrain is reasonably level the map plotting of objects can be more rapidly accomplished by the use of a *Canadian Grid*. Detail of grid mapping is available in many textbooks of photogrammetry as, for example, *The Manual of Photogrammetry*, issued by the American Society of Photogrammetry, Washington, D. C.

A new method overcomes the problems of extracting molybdenite from Morenci copper concentrates.

Flotation of Molybdenite At the Morenci Concentrator

by J. E. Papin

MORENCI ores contain as an average about 0.015 pct molybdenite, MoS_2. Incidental to the concentrating operations applied for the recovery of copper minerals, approximately two-thirds of the molybdenite is floated and appears in the final copper concentrate. The economic importance of molybdenum and the success achieved in its recovery in milling operations elsewhere encouraged research directed toward its recovery in marketable form.

Several procedures are utilized to effect separation of molybdenite from copper and iron sulphides, but only two have wide application. In the better known of these two methods the molybdenite in the copper concentrate is depressed in a flotation operation using soluble starch as the depressant. The tailing from this treatment thus becomes a low-grade molybdenite concentrate which after thickening, filtering, and low temperature roasting is repulped with water and subjected to additional flotation steps for recovery of the molybdenite. A thio-phosphate collecting agent is employed in flotation of the copper minerals. Presumably the stability and lasting effects of that collector type necessitated the practice of depressing molybdenite followed by subsequent roasting to insure elimination of copper and iron sulphides from final molybdenite concentrate.

The second important method of recovery is applied at various southwestern plants. In those plants xanthates are in use as copper collecting agents. It has been found that xanthates are relatively unstable and their collecting power for sulphides is destroyed by very simple procedures. The method generally applied is to subject the xanthate-activated concentrate, in the form of a pulp, to prolonged heating using steam as the heating medium. A concentrate thus treated may then be subjected to a flotation operation for recovery of the molybdenite, using mineral oil collecting agents, and the copper and iron sulphides will be depressed.

Neither of the above processes was applicable to Morenci concentrates. Morenci uses a thio-phosphate collecting agent in its flotation operation. However, it was established that soluble starch was not a depressant for Morenci molybdenite; therefore the first step in that process was not applicable. As would be expected the copper and iron sulphides in Morenci concentrates could not be depressed by the heating methods used on xanthate activated concentrates. Low-temperature roasting did eliminate the effects of the thio-phosphate but had to be rejected as uneconomic in as much as the tonnage of concentrates involved was too great.

Preliminary research having established the above facts, it became obvious that new methods would have to be developed for the recovery of molybdenite from Morenci concentrates. A broad laboratory investigation was initiated along two general lines, both directed toward depression of copper and iron sulphides. One approach was through the use of water-soluble oxidizing agents with the objective of destroying the thio-phosphate in the concentrate and thus eliminating its collector effect. The other approach involved the use of soluble sulphides, which have long been recognized as sulphide depressants. These efforts resulted in laboratory processes which showed promise. Research was then extended to a pilot plant having a daily capacity of several tons of concentrate. This operation soon demonstrated that soluble sulphide—basically sodium polysulphide—when applied to a pulp made slightly acid with sulphuric acid, proved to be the most effective sulphide depressant. The rate of concentrate treatment in this plant and the low molybdenite content of the concentrate prevented carrying the process to a conclusion, namely, production of molybdenite concentrate of marketable grade. Laboratory treatment of pilot plant concentrate indicated that a marketable product could be made from it, given a tonnage suited to available cleaner flotation machine capacity. Accordingly, the molybdenite plant was enlarged to permit treatment of the total copper concentrate from the extension plant, maximum of approximately 800 tons per day.

In the enlarged plant a major difficulty was encountered. No criteria could be established as a basis for control of the process. In the early stages of flotation, because of the small amount of molybdenite involved, there was no visual evidence of the relative floatability of molybdenite with respect to

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Discussion on this paper, TP 3954B, may be sent (2 copies) to AIME before April 30, 1955. Manuscript, Sept. 8, 1954. San Francisco Meeting, September 1954.

The Transactions papers appearing in Mining Engineering during 1955 will subsequently appear in AIME Transactions Vol. 202 and may be permanently referenced to that volume.

other sulphides and therefore no evidence of the depressing effects being applied by soluble sulphide additions. In other words, the operation could not be so stabilized that maximum recovery of molybdenite could be achieved simultaneously with a proper grade of concentrate.

Table I. Point of Addition and Amounts of Reagent Used

Sulphuric Acid	Original Feed, Lb Per Ton
Conditioner ahead of roughing flotation	1.46
No. 3 conditioner ahead of primary cleaning flotation	0.46
No. 2 conditioner ahead of secondary cleaning flotation	0.40
Surge box ahead of primary recleaners	0.06
Total	2.38
Sodium Ferrocyanide	
Discharge of conditioner ahead of roughing flotation	0.30
Discharge of conditioner ahead of primary cleaning flotation	0.22
Discharge of conditioner ahead of secondary cleaning flotation	0.11
Pump sump, feed to tertiary cleaning flotation	0.08
Feed box to primary recleaning flotation	0.04
Total	0.75
Sodium Cyanide	
Feed to final Denver recleaning flotation cell	0.61
Sodium Polysulphide	
No. 2 conditioner ahead of secondary cleaning flotation and in regrind ball mill	0.07

To guide plant operation and to search for better means of effecting molybdenite recovery, laboratory work was continued throughout this period of process development. Among other depressants studied were the ferro and ferri-cyanides. Gaudin in 1932[1] had cited them as depressants for chalcocite, and recently their application in a sphalerite-chalcocite separation has been described in the technical press. Both the ferro and the ferri-cyanides proved to be excellent depressants of the copper and iron sulphides when applied in an alkaline pulp of low pH. There was no evidence of depression of molybdenite at any concentration of the cyanides and molybdenite recovery was therefore relatively good. Quantities required were within economic limits and acid requirements were reduced as compared with the soluble sulphide process. More important was the fact that with their application there was marked visual evidence of differential flotation to serve as a criterion for the control of reagent additions. Their prompt application in the molybdenite plant was therefore undertaken, see Table I.

Sodium ferro-cyanide, being the least costly salt, was selected for initial plant use. It was first applied in the roughing flotation step and after proving satisfactory there was also used in several cleaning steps of the process. However, in the final cleaning steps depression of copper and iron sulphides was not adequate and the ferro-cyanide was replaced by sodium cyanide. The process as finally developed involves application of pulp density and pH control, methods of reagent addition, and operating techniques which, taken together, are believed to represent a contribution to the art. The important process features will be presented in more detail in the discussion of the plant flowsheet, Fig. 1. See also Tables II and III for metallurgical data.*

* U. S. patents 2,559,104; 2,608,298; and 2,664,199 have been issued to Phelps Dodge Corp. on processes described in this paper.

Copper concentrate is sent to the molybdenite rougher flotation along with the molybdenite primary cleaner tailing. This pulp is pumped to a conditioner where water is added for dilution and sulphuric acid is added for pH control.

Pulp from the conditioner is diverted to the rougher flotation units and controlled addition of ferro-cyanide is made at the feed inlet to the first cell of each flotation bank. Effects of the ferro-cyanide are relatively short-lived owing, presumably, either to consumption of the reagent through reaction with sulphides or through its destruction by chemical constituents of the pulp. Long conditioning of the pulp after ferro-cyanide addition must therefore be avoided. Addition of ferro-cyanide at this point is limited to a certain amount so that some flotation of copper and iron sulphides will occur in the last flotation cells in each bank. Tailing from these cell banks is the final tailing of the plant and is diverted to a thickener.

The molybdenite rougher concentrates pass to a thickener where they are joined by cleaner tailing from the secondary and tertiary cleaners. Thickening is to accomplish partial elimination of frothing agents which have carried over from the copper plant and to provide controlled feeding of the concentrate to the molybdenite plant primary cleaners. It was established by initial laboratory and pilot plant work that control of froth formation was essential in all the molybdenite flotation steps. Factors in this control were the amount of residual frothing agent in the concentrate feed to each flotation unit and the extent to which copper and iron sulphides were depressed. With a maximum of frothing agent present, prohibitive amounts of depressant were required to prevent flotation of copper and iron sulphides. In furtherance of the principle control of froth formation the thickened concentrates are delivered to a conditioner where fresh

Table II. Molybdenum Plant, General Data

Description	Pulp Data: Solids, Pct	Pulp Data: Sp Gr	Pulp Data: pH	Dry Ore: Sp Gr	Dry Ore: Average, Tons Per 24 Hrs
Feed to molybdenum plant roughers	30.0		11.0	5.2	1500
Pulp in conditioner before rougher flotation	25.0	1.253		5.2	3300
Rougher flotation	25.0	1.253	7.0 to 8.0	5.2	3300
Underflow, 100-ft thickener	50.0	1.667		5.0	1500
Conditioner ahead of primary cleaner flotation	25.0	1.250		5.0	1500
Primary cleaner flotation	25.0	1.250	7.0 to 8.0	5.0	1500
Conditioner ahead of secondary cleaner flotation	10.0	1.089		5.4	335
Secondary cleaner flotation	10.0	1.089	7.0 to 8.0	5.4	335
Tertiary cleaner flotation	10.0	1.087		5.0	80
Underflow, 20-ft thickener	40.0	1.463		4.8	40
Primary recleaner flotation	2.5	1.020	7.5 to 9.0	4.8	40
Final recleaner flotation	2.0	1.016	9.0 to 11.5	4.6	9

Sulphuric acid and sodium ferrocyanide are used in all flotation stages except the final cleaner flotation stage, at which point sodium cyanide is used.

water is added for dilution, and sulphuric acid is added again for pH control. Ferro-cyanide in second stage is added to the feed inlets of the first cell of each of the flotation banks. Continuous pH control is held at a pH of 7.5 with an automatic controller.

Concentrates pass to a conditioner and from there to the secondary cleaner cells. The tailing is delivered to the conditioner ahead of the rougher flotation units. Acid is added to the conditioner ahead of the secondary cleaners, and ferro-cyanide to the feed inlet of the first cell.

The concentrate made by the tertiary cleaners is delivered to a thickener and the tailings are recycled through the primary cleaning step. Thickening of the concentrate is necessary for two reasons: 1—to eliminate frothing agents which have been concentrated by this and preceding flotation steps, and 2—to prepare the product for filtration and subsequent grinding. A considerable portion of the molybdenite is locked with or attached to gangue material. If it is not freed from the gangue it is lost during subsequent treatment or if recovered carries the gangue with it, with the result that the final concentrate is contaminated.

The reground concentrate is diluted to desired density with fresh water, acid is added to adjust the pH, and ferro-cyanide is added immediately prior to introduction of the pulp into the four-cell primary recleaning unit. A minor amount of frothing agent may be necessary in this step to maintain froth production. The concentrate obtained is diverted to the final cleaner unit and the tailings are routed to the copper concentrate thickener.

The final cleaner unit consists of four flotation cells in stages. The feed to the unit enters the first cell of the series. Concentrate from that cell passes to the second cell, concentrates from the second cell advance to the third cell, and so on to the fourth cell, where final concentrate is produced. There are thus six stages of cleaning involved. Concentrated sodium cyanide solution is fed into the fourth cell and moves with the tailings from the several cells countercurrent to the concentrate flow. Tailing from

Table. III. Assays and Distribution of Feed, Concentrate, and Tailing, January Through May, 1954

Item	Wt Pct	Assays MoS₂	Assays Cu	Assays Fe	Distribution, Pct MoS₂	Distribution, Pct Cu	Distribution, Pct Fe
Heading (computed)	100.00	0.284	22.63	27.91	100.00	100.00	100.00
Roughing tailing	99.80	0.106	22.58	27.94	37.25	99.61	99.90
Final concentrate	0.20	89.09	0.44	1.4	62.75	0.39	0.10
Ratio of concentration	499.91						

Assays of Intermediate Concentrates, Special Samples Taken One Day Only	Assay MoS_2	Assay Cu
Molybdenite plant feed	0.156	20.88
Roughing concentrate	0.489	36.35
Primary cleaning concentrate	4.22	52.67
Secondary cleaning concentrate	14.50	44.31
Tertiary cleaning concentrate	29.31	34.44
Primary recleaner concentrate	72.95	6.20
Final concentrate	85.06	0.40

this series of cleaning steps exists from the first cell of the bank and is returned to the thickener. It is to be noted that in this operation the molybdenite-rich material is advanced progressively through an environment of increasing cyanide concentration. This is essential to eliminate a maximum of the copper and iron sulphides which have escaped depres-

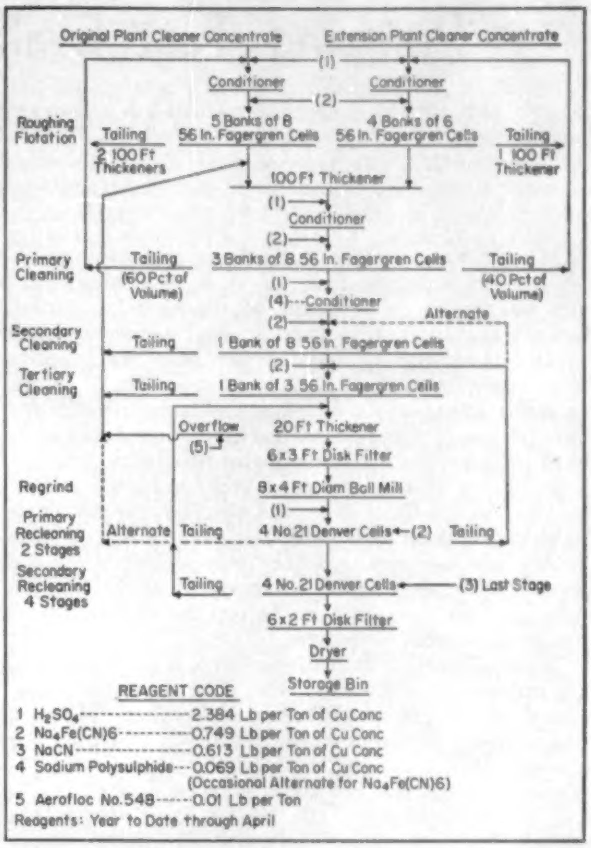

Fig. 1—Flotation of molybdenite at the Morenci concentrator, May 1954.

sion in preceding operations. The amount of sodium cyanide addition is determined on the basis of froth appearance and periodic analytical determination of the copper content of the finished concentrate. In usual flotation practice, cleaner tailings are returned directly to a preceding flotation step. In this instance the sodium cyanide addition has raised the pH of the cleaner tailing to such an extent that its return to a preceding circuit would complicate pH control in that circuit. It is therefore diverted to the concentrate thickener, where dilution followed by partial dewatering is accomplished, and joins the new feed to the plant.

Significant features of this molybdenite flotation process may be briefly summarized as follows: 1—Dewatering of rougher concentrate followed by subsequent dilution with fresh water partially to eliminate frothing and collecting agents and permit control of froth formation. 2—Regulation of pulp density with the same objective. 3—Control of pH to provide proper environment for the functioning of the depressant sodium ferro-cyanide. 4—Step addition of the ferro-cyanide to maintain maximum depressing effects on copper and iron sulphide with a minimum quantity of the reagent. 5—Final cleaning of the molybdenite concentrate by use of a powerful depressant, sodium cyanide, under conditions designed to minimize reagent requirements.

Acknowledgment

The writer expresses thanks to the Management of Phelps Dodge Corp. for permission to publish this paper.

Reference

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Pumping Test Evaluates Water Problem At Eureka, Nev.

by Wilbur T. Stuart

TO assist the mining industry in attacking problems of water control, the U. S. Geological Survey has begun a program of research in mining hydrology. In certain fundamental respects water control is similar to development of water supplies from wells or to the drainage of agricultural lands, as many of the tools developed in recent years for quantitative ground-water problems are applicable, with modification, to mine-water problems.

In 1952 a 30-day pumping test conducted jointly by the Eureka Corp. Ltd. and the Defense Minerals Exploration Agency provided an opportunity to gain knowledge concerning water movements around a flooded mine shaft. The methods of analyzing the data may be used as a guide for the evaluation of similar problems elsewhere.

The Fad shaft of the Eureka Corp. is on Ruby Hill, 1½ miles west of Eureka, Nev. The shaft was completed at a depth of 2465 ft in November 1947 at a site adjacent to the downfaulted block in which the ore was found. As the drift on the 2250 level progressed toward the ore zone, a large flow of water was encountered after the Martin fault was intersected. This flow exceeded the installed pump capacity, and an unsuccessful attempt to recover the shaft and the 2250 level was made in 1948.[1,2]

Geology and Hydrology: The complex structure of Ruby Hill is that of an anticline broken first by thrust faulting and later by normal faults. The present orebody comprises several mineralized zones within a block of the Eldorado limestone of middle Cambrian age which was downfaulted 1400 to 1600 ft, and it may be related to a similar body mined at a higher level south of the Ruby Hill fault. At the depth of the largest zone of ore the block is roughly rectangular in shape, about 1000 ft wide and 1500 ft long, and dips about 30° NE, see Fig. 1. It is apparently bounded on the south by the Ruby Hill fault, on the east by the Jackson fault, on the north by the Martin fault, and the west by the Bowman fault. Within the block, but between the Ruby Hill and the Martin faults, are the Office and Adams Hills faults; west of the block and the Bowman fault are the Albion and Spring Valley faults.

There are many conflicting reports concerning the water-yielding characteristics of the Eldorado limestone and the condition of the fault zones, that is, whether they are open or tight. However, the diamond-drill records indicate that open spaces as much as 2 or 3 ft across were encountered, and considerable cementing and lining of holes was necessary to maintain circulation of drilling fluid. There is also evidence that the Eldorado limestone was cavernous where it was mined in the early days south of the Ruby Hill fault.

At the site of the Fad shaft the formations encountered from the surface down included the Pogonip limestone, Dunderberg shale, Hamburg limestone, and Secret Canyon shale. These formations did not yield large quantities of water to the shaft. The Pogonip limestone, which appears to be permeable and might yield water elsewhere, is above the water table in the vicinity of the shaft. The Secret Canyon shale, immediately overlying the Eldorado in some places but in most places separated from it by the Geddes limestone,[3] is apparently tight and does not transmit water. During the 30-day test period the shale briefly confined the water in the underlying formations so that artesian conditions were observed in drillholes *E* and *F*, which are cased into the Eldorado limestone, whereas unconfined conditions were observed in drillholes *B*, *C*, and *D*, which were open to the shale. The Geddes limestone, which normally lies between the Secret Canyon shale and the Eldorado limestone, was not encountered in the Fad shaft. The Geddes, a flaggy, fractured limestone, is reported capable of yielding large volumes of water. Water stored in the interstices of this thin-bedded limestone within the Ruby Hill fault zone on the 1200 level of the Locan shaft drowned the pumps in 1923 when the Richmond-Eureka Mining Co. attempted to explore the area along the fault. Eldorado limestone was not encountered in the Fad shaft.

In the ore-block area the Eldorado limestone was not entirely offset from other water-yielding formations by movement along the Bowman fault; therefore it may be hydraulically connected with the other formations. Adjacent to the Ruby Hill, Jackson, and Martin faults, the Eldorado lies in contact with other possible water-yielding formations. One of these, the Prospect Mountain quartzite, is separated from the Eldorado by thin, sheared, and broken beds of the Geddes within the Ruby Hill fault zone.

A limited examination by the author of the Prospect Mountain quartzite in the Richmond mine at a higher level and south of the Ruby Hill fault indicates that the quartzite is poorly permeable. The monzonite mass south of the quartzite would be a further barrier to the flow of water. The poor permeability of this area is substantiated by records of levels at which water was encountered south of the Ruby Hill fault. In view of the normally low rate of ground-water recharge, if this desert area had been permeable, water levels could not have been maintained at altitudes of many hundred feet above the present water table west and north of Ruby Hill.

Thus the ore-bearing block of Eldorado limestone is in contact with possible water-yielding rocks on at least two sides, and if the fault zones are possible conduits for water circulation the geologic and hydrologic conditions are suitable for the infinite-aquifer type of analysis as used and modified here.

History of Pumping: During sinking of the Fad shaft a maximum pumping rate of 1500 gpm kept the shaft dewatered sufficiently, but in March 1948, after the 2250 level drift passed through the Martin fault into the Eldorado limestone, the pumps and shaft were flooded. Subsequently additional pump-

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Discussion on this paper, TP 3967A, may be sent (2 copies) to AIME before April 30, 1955. Manuscript, June 1, 1954. New York Meeting, February 1954.

ing equipment was installed and over a period of several months the pumping rate was increased to 8000 gpm. Pumping at the 8000 gpm rate lowered the water level in the shaft to within 170 ft of the 2250 level. Increasing the rate of pumping from 8000 to 9000 gpm came within 60 ft of recovering the 2250 level, but this high rate of pumping began to bring gouge and pebbles of limestone into the shaft and the water became very muddy. Rate of pumping was immediately reduced to about 7500 gpm and the water became clear. At the time the water from the shaft became muddy, the pumping level rose approximately 340 ft in the shaft, and although the pumping rate was later increased to about 9000 gpm the water level in the shaft remained about 400 ft above the 2250 level. It is probable that much fine sediment was removed from the fractures as the water moved along the Martin fault. As a result it appears that there is a better hydraulic connection between the main block of the Eldorado limestone and the shaft than before the shaft was pumped at 9000 gpm.

It is estimated that by Nov. 14, 1948, 30 tons of material were pumped from the mine and an additional 108 cu yd had settled in the bottom of the Fad shaft. By Nov. 23, 1948, soundings indicated that about 360 cu yd of the material had settled in the bottom. It was feared that eventually the shaft would fill above the 2250 level, from which the drift to the ore zone has been started.

During attempts to recover the 2250 level, depth-to-water measurements were made in drillhole *E*, which taps the ore-bearing formations in the Eldorado. Although the water level in drillhole *E* was lowered about 330 ft during these periods of pumping, it was still about 880 ft above the 2250 level. This indicated not only that the water table had a shape of an inverted cone with the apex at the shaft, but that recovery of the 2250 level at the shaft would not unwater the orebody.

In the period of operations from March 1948 to December 1948 about 5000 acre-ft of water was pumped from the shaft. Measurements of the altitude to which the water level in the drillholes recovered after pumping stopped in 1948 show little if any permanent lowering of the water level in the ore zone. Rough calculations show that if all this water had been removed from storage in the ore-bearing block, the level in the ore zone should have been lowered 250 to 450 ft. Thus it appears that a large part of the water is moving into the ore zone, either from other formations in contact with the Eldorado or through fault zones surrounding the ore-bearing block.

Pumping Test: To evaluate the feasibility of unwatering the shaft and the associated ore zone, a pumping test was made to determine the ability of the formations to transmit water to the shaft and ore-zone area. For 30 days the shaft was pumped at 3600 gpm and the rate of lowering of the water level was observed in the shaft and at all other available points; the rate of rise after pumping ceased was observed for 10 days. Normally the recovery period should be equivalent to the drawdown period, but in this case the data indicated that the recovery was duplicating the information obtained in the drawdown test. The recovery period was shortened, therefore, to facilitate operations in unwatering the property. Two 450-hp submersible pumps lifted the water from the shaft to the 789-level sump. Opening or closing valves in the pump-discharge line maintained a constant head on an orifice, insuring uniform flow rate. Water from the 789-level sump was lifted to the collar of the shaft by means of 1000-gpm station pumps and passed through a Parshall flume equipped with an electrical recorder, which provided a graphic record of flow rate and total flow in thousand gallons. A continuous graphic record of the depth to water in the shaft was obtained by means of a float-operated instrument checked manually three times each day. Depth to water in drillhole *E* was recorded by means of a similar instrument. Depth to water in drillholes *B*, *C*, *D*, and *F* and in the Locan shaft was measured manually as often as time permitted. A float-operated recording instrument was installed in the Holly shaft about 6500 ft north of the Fad shaft to determine changes in the water level at that point. Depth-to-water measurements were made manually at weekly intervals or oftener in four drilled and dug stock wells in Diamond Valley at distances of 3 to 6½ miles north of the Fad shaft. All measurements were made to the nearest 0.01 ft.

Pumping in the Fad shaft began at 9:53 am on Dec. 23, 1952, and continued at the uniform rate of 3600 gpm until 9:06 am on Jan. 22, 1953. After pumping ceased the altitude of the water levels was determined at periodic intervals in each of the diamond drillholes and in the Locan shaft for a period of 10 days. Graphs of the drawdown and recovery of water levels at each site are shown on Figs. 2 and 3. Total drawdown and recovery, in feet, for each of the sites in which depth to water measurements were made are shown in Table I.

Table I. Total Drawdown or Recovery of Water Level, in Feet, During Test Period

	Drawdown	Recovery
	Dec. 23, 1952, to Jan. 22, 1953	Jan. 22, 1953, to Feb. 1, 1953
Fad shaft	264.66	239.98
Drillhole *B*	172.16	Hole plugged during recovery
Drillhole *C*	100.15*	17.87
Drillhole *D*	187.75	110.60
Drillhole *E*	101.34	82.75
Drillhole *F*	97.19	79.24
Locan shaft	91.30	73.23
Holly shaft	Not affected by pumping	
Airport well 20/53-15bl	Not affected by pumping	
Rodeo ground well	Not affected by pumping	
SW Diamond Valley well	Not affected by pumping	
A. C. Florio well 19/53-5al	Not affected by pumping	

* Decline of water arrested at this point, probably as a result of caving of shale.

Analysis of Pumping Test: Withdrawal of water from any permeable material causes the water levels to decline in the vicinity of the point of withdrawal, and the shape of the water table or piezometric surface becomes an inverted cone, the apex at the point of withdrawal. In some cases, as in a mine shaft, the shape of the cone is modified or elongated to fit the hydrologic characteristics of the rock structure, the openings into the formation, and the boundaries. However, the overall size, shape, and rate of growth of the cone of depression caused by pumping the shaft are dependent on rate and duration of pumping; transmissibility and storage coefficients of the formations; increase in recharge, if any, induced by the declining water level; natural discharge that is salvaged; and boundaries of the bedrock formations.

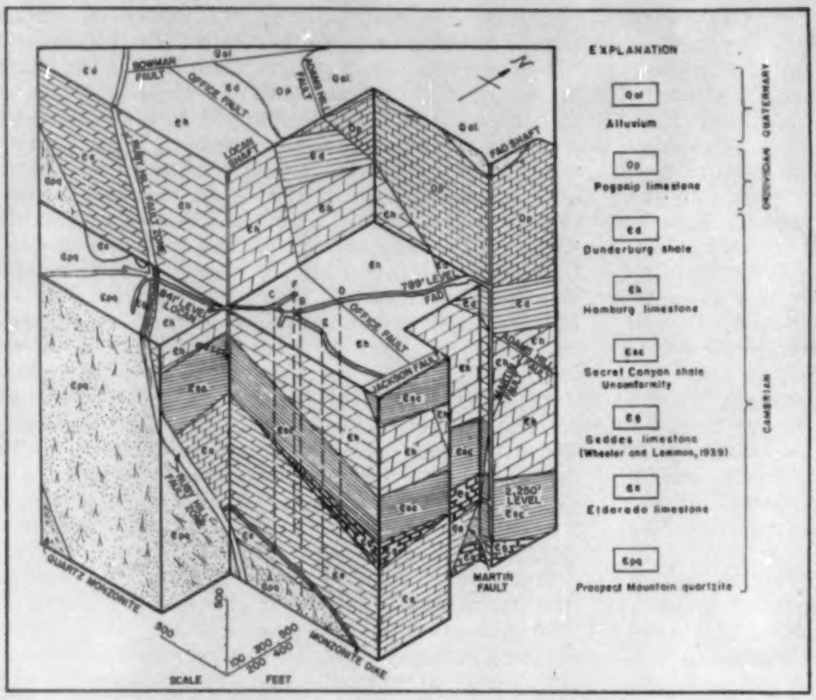

Fig. 1—Block diagram shows formation and location of drillholes, Locan and Fad shafts, Eureka Corp. Ltd.

The lowering at any point of the cone of depression is termed *drawdown* and is dependent on the above variables and distance from point of withdrawal.

To express in a general equation the relationship among the variables that governs the magnitude of the unwatering, certain basic assumptions are made. It is assumed that the aquifer is constant in thickness, infinite in areal extent, and homogeneous and isotropic, that is, transmitting water with equal facility in all directions. It is assumed further that the water may enter the shaft from the full thickness of the aquifer.

The relationship among the hydraulic properties of a formation, the rate of pumping, and the change in water level caused by withdrawal of water from a well or shaft in a given formation is expressed by the following formula developed by Theis:[4]

$$s = \frac{114.6\,Q}{T} \int_{\frac{1.87\,r^2 S}{T\,t}}^{\infty} \frac{e^{-u}}{u}\,du$$

where s = drawdown, in feet at any point in the vicinity of a shaft pumped at a uniform rate; Q = discharge of shaft, gpm; T = coefficient of transmissibility, gpd per ft; r = distance, in feet, from pumped shaft to point of observation; S = coefficient of storage; t = time, in days, that shaft has been discharging; and $u = 1.87\, r^2S/Tt$.

To use the formula to determine the effect on the water level, at any distance from a shaft, caused by a change in rate of withdrawal it is necessary to know the water-bearing characteristics of the formations. These characteristics are called the coefficients of transmissibility and storage. Transmissibility may be defined as the number of gallons of water that will move per day through a vertical strip of the aquifer 1 ft wide with a height equal to the full thickness of the aquifer when the hydraulic gradient is reduced 1 ft for each foot of water travel through the formation. Coefficient of storage is defined as the volume of water, measured as a fraction of a cubic foot, released from storage in each column of the aquifer having a base of 1 sq ft and a height equal to the thickness of the aquifer when the head is lowered 1 ft.

The hydraulic characteristics of the undisturbed water-bearing formations in place are determined by pumping tests. A pumping test is made by changing the rate of withdrawal and observing the effect on the drawdown or recovery of the water levels in other parts of the cone of depression. By analysis of these data, values of the transmissibility and storage coefficients can be obtained. Stated in another way, a pumping test determines the equation of the drawdown curve so that the curve may be extended by means of the same equation for a longer period of time and for other distances and rates of pumping. When the equation is applied for

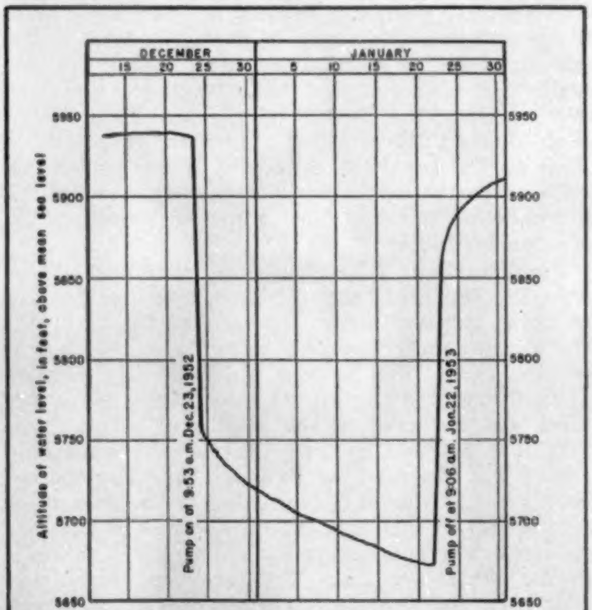

Fig. 2—Drawdown and recovery of water level in the Fad shaft during test period.

a longer period, consideration must be given to the effects caused by boundaries and changes in the character of the formation. Simple boundaries of the type through which water cannot pass increase the rate of drawdown, whereas boundaries of the type which add water to the system reduce the rate. When predictions based on mathematics are made it is assumed that the aquifer is infinite, and hypothetical image wells are introduced at an equal distance on the opposite side of the boundary, discharging an amount equal to that pumped or recharging a similar amount, depending, respectively, upon whether the boundary is impermeable or water is added. A boundary of the barrier type, through which no water can pass, was indicated in analysis of the drawdown curves for the Fad shaft and drillholes *E* and *F*. This boundary, approximately 1 mile from drillhole *E*, was noted during the test, 20 days after pumping started. No boundaries were noted of the type indicating that water (outside recharge) was added to the hydraulic system. A general conclusion may be made from the alignment of drawdown and recovery curves: the hydraulic system feeding water to the Fad shaft and the orebody reaches extensively in three directions, and within the 30-day period of test no large outside source of water, excluding that already in storage within the bedrock, was encountered within 1.1 miles of the Fad shaft.

Table II. Coefficients of Transmissibility and Storage of the Bedrock as Computed from Pumping Tests on the Fad Shaft

	Drawdown		Recovery	
Shaft or Drillholes	Coefficient of Transmissibility, Gpd Per Ft	Coefficient of Storage	Coefficient of Transmissibility, Gpd Per Ft	Coefficient of Storage
Fad shaft with barrier-type boundary after 15 days	16,600		16,100	
Drillhole *E* with barrier-type boundary after 20 days	23,800	0.00074	21,100	0.00066
Drillhole *F* with barrier-type boundary after 22 days	24,000	0.00066	21,300	0.00064
Locan shaft	16,100	0.0014	15,420	0.00118

In the pumping test at Eureka transmissibility and storage coefficients have been determined by the methods of Theis[4] and Cooper and Jacob.[5] Both methods were used to analyze the pumping-test data, as each has advantages not possessed by the other.[6] Both methods were applied to the drawdown and to the recovery of the water levels observed in the Fad shaft, in drillholes *E* and *F*, and in the Locan shaft. Computations are shown on Figs. 4-12. The drawdown and recovery of the water level in drillholes *B*, *C*, and *D* showed the effects of unwatering formations above the Secret Canyon shale; these methods cannot be rigorously applied, as these holes do not penetrate the formation fully and the saturated thickness in the holes was materially reduced during the test.

Values of the formation constants obtained during the test are shown in Table II.

Evaluation of Pumping Problems: A method of application of the hydraulic characteristics of the formation is proposed in the following example to determine how much water must be withdrawn to unwater the shaft and ore zone within a scheduled period.

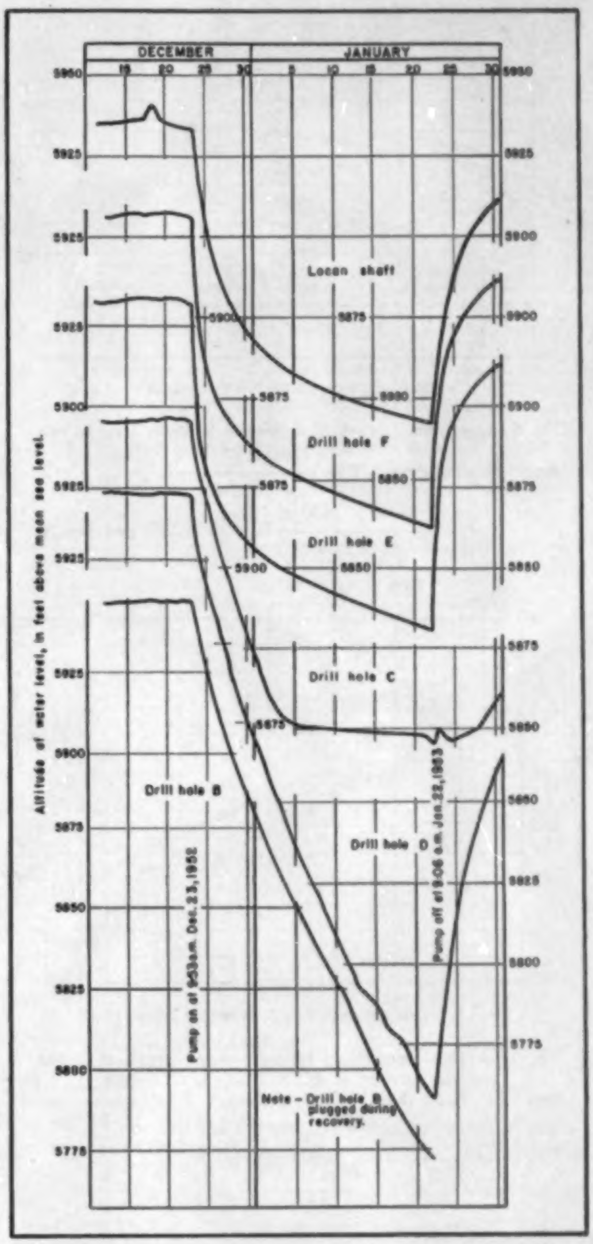

Fig. 3—Drawdown and recovery of water levels in drillholes B, C, D, E, and F and Locan shaft during test period.

The nonequilibrium method developed by Theis makes it possible to compute the drawdown or recovery at any point, at any time, and for any rate and period of pumping. It assumes that transmissibility and storage coefficients remain constant throughout the aquifer and during the computed period. These conditions were not completely fulfilled during the pumping test, as transmissibility is greater in the vicinity of the ore zone than in the neighborhood of the shaft. The effective radius of the shaft varies, owing to irregularities in bore and in the volume of the levels unwatered. Nevertheless, in spite of the irregularities, the calculations can be made without difficulty by averaging of variations for each factor.

Because unwatering of the Fad shaft will progress under the same conditions existing when it was tested, a simple graphic method is proposed which automatically includes changes in transmissibility and storage, boundary conditions, and irregularities

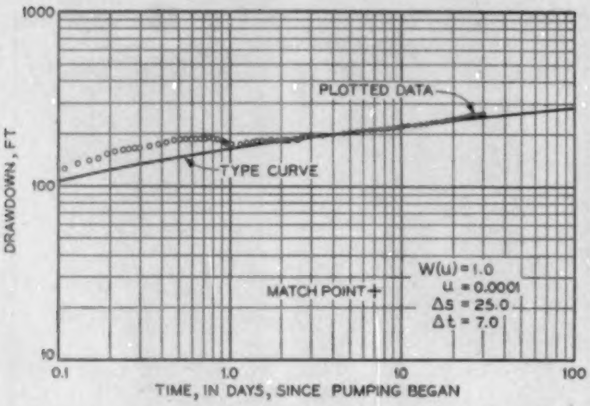

Fig. 4—Logarithmic graph of drawdown of water level in Fad shaft. Computations: $T = \dfrac{115.6\,Q\,W(u)}{\Delta s}$,

$$T = \frac{114.6 \times 3600 \times 1.0}{25.0} = 16{,}500 \text{ gpd per ft.}$$

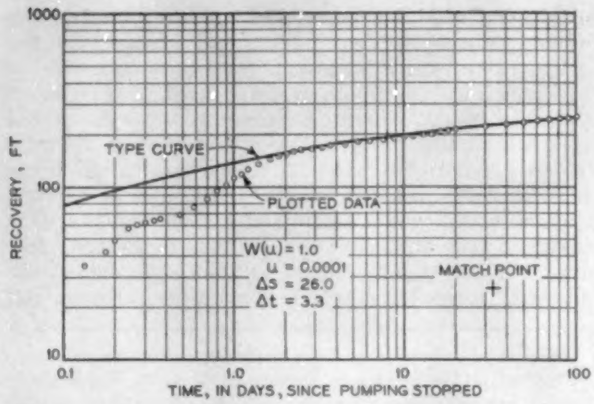

Fig. 6—Logarithmic graph of recovery of water level in Fad shaft. Computations: $T = \dfrac{114.6\,Q\,W(u)}{\Delta s}$,

$$T = \frac{114.6 \times 3600 \times 1.0}{26.0} = 16{,}000 \text{ gpd per ft.}$$

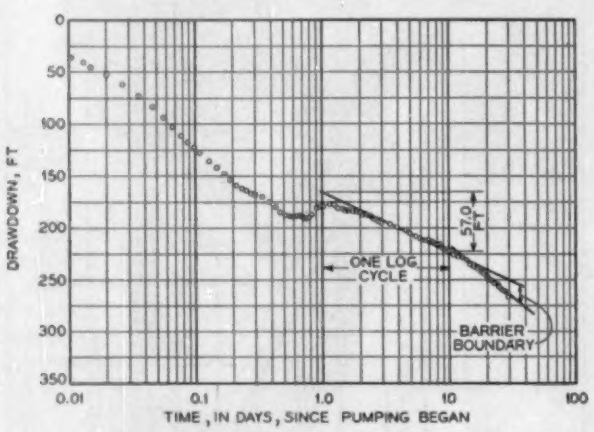

Fig. 5—A semi-logarithmic time-drawdown graph of water level in Fad shaft. Computations: $T = \dfrac{263.5\,Q}{\Delta s}$,

$$T = \frac{263.5 \times 3600}{57.0} = 16{,}600 \text{ gpd per ft.}$$

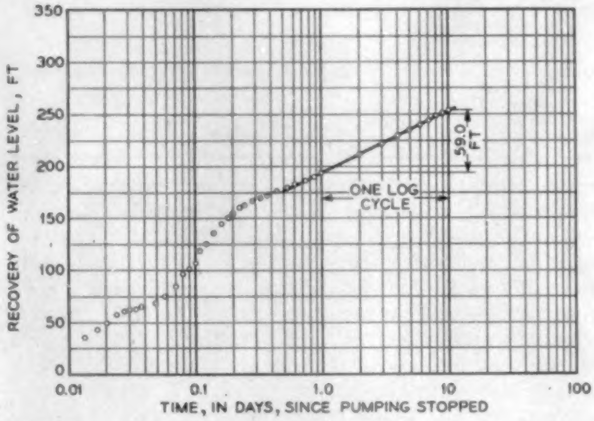

Fig. 7—A semi-logarithmic time-recovery graph of water level in Fad shaft. Computations: $T = \dfrac{263.5\,Q}{\Delta s}$

$$T = \frac{263.5 \times 3600}{59.0} = 16{,}100 \text{ gpd per ft.}$$

in the shaft. Fig. 13 is a time-drawndown graph for a pumping rate of 1000 gpm at the Fad shaft. In as much as drawdown is proportional to rate of withdrawal, the graph was constructed by use of 1/3.6 of the measured drawdown occurring during the 30-day test. After this period the drawdown is extrapolated for a transmissibility of 16,500 gpd per ft with one impermeable boundary. The drawdown for any other pumping rate is in direct proportion to that rate.

Observations in 1948 and in the 1952 test showed that pumping from the shaft lowers the water level in the ore zone at the sites of holes *E* and *F*. For convenience the site of hole *E* has been selected for determination of the amount of lowering. Fig. 14 is a time-drawdown graph showing the drawdown that will occur at the site of drillhole *E* caused by the pumping of 1000 gpm at the Fad shaft 980 ft away. It was constructed on the basis of 1/3.6 of the measured drawdown during the 30-day test period and extrapolated by the use of a coefficient of transmissibility of 24,000 gpd per ft with one impermeable boundary.

Construction of Figs. 13 and 14 is based on the assumption that the bedrock hydrology will not change materially as the cone of pumping expands outward beyond the limits considered in the 30-day test. It is further assumed that any intersected boundaries of one type will be balanced by the intersection of the same number of boundaries of the opposite type, in other words, the rock in the surrounding area will act as an infinite aquifer, except for modification caused by the one boundary.

Example of Method of Calculation for Time and Rate for Unwatering Shaft: The following calculations are made to demonstrate the method and should not be considered the most economical or recommended rates, as these must be determined by good mining practice. In all calculations the average rate of pumping must be continued for the full length of time and each successive increase or decrease in the rate must be considered as a separate increment superimposed on the preceding system. For example, if a system has been pumped at the rate of 1000 gpm for 2 years, and the rate is increased to 2000 gpm, it is desired to know the drawdown at the end of 3

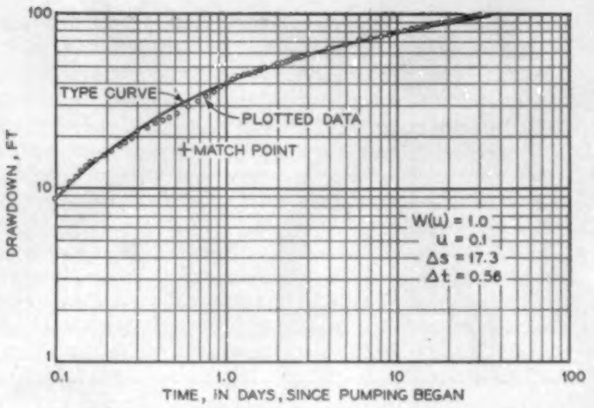

Fig. 8—Logarithmic graph of drawdown at hole *E* caused by pumping at Fad shaft. Computations: $T = \dfrac{114.6\,Q\,W(u)}{s}$,

$$T = \frac{114.6 \times 3600 \times 1.0}{17.3} = 23{,}900 \text{ gpd per ft},$$

$$S = \frac{T\,u\,\Delta t}{1.87\,r^2},\quad S = \frac{23{,}900 \times 0.1 \times 0.56}{1.87\,(980)^2} = 0.00075.$$

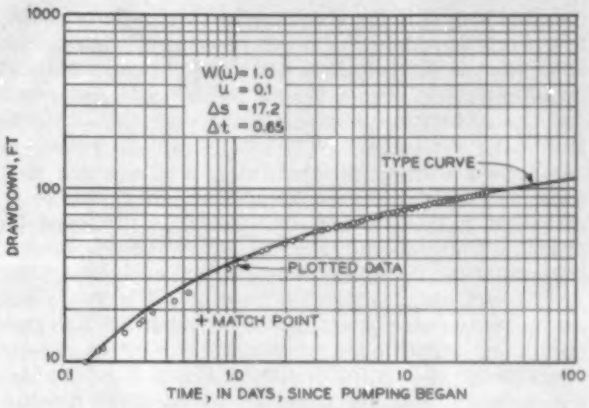

Fig. 10—Logarithmic graph of drawdown in hole *F* caused by pumping at Fad shaft. Computations: $T = \dfrac{114.6\,Q\,W(u)}{\Delta s}$

$$T = \frac{114.6 \times 3600 \times 1.0}{17.2} = 24{,}000 \text{ gpd per ft},$$

$$S = \frac{T\,u\,\Delta t}{1.87\,r^2},\quad S = \frac{24{,}000 \times 0.1 \times 0.65}{1.87\,r^2} = 0.00069.$$

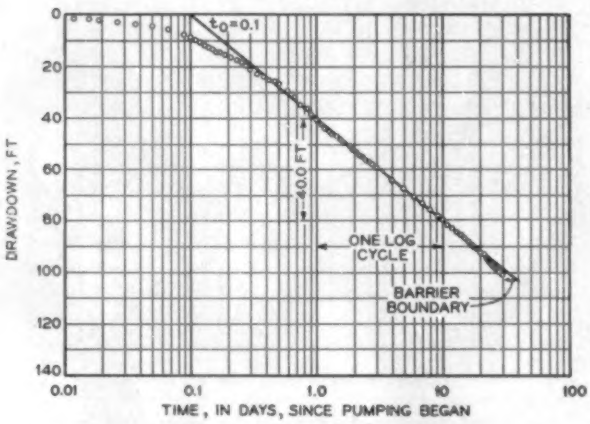

Fig. 9—A semi-logarithmic time-drawdown graph of interference of Fad shaft pumping on hole *E*. Computations:

$$T = \frac{263.5\,Q}{\Delta s},\quad T = \frac{263.5 \times 3600}{40.0} = 23{,}800 \text{ gpd per ft},$$

$$S = \frac{0.301\,T\,t_0}{r^2},\quad S = \frac{0.301 \times 23{,}800 \times 0.1}{(980)^2}.$$

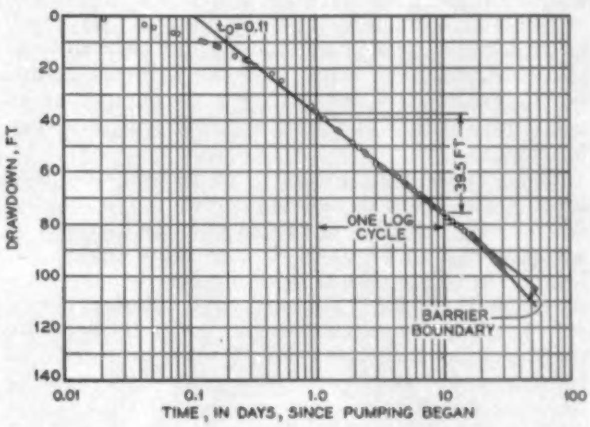

Fig. 11—A semi-logarithmic time-drawdown graph of interference of Fad shaft pumping on hole *F*. Computations:

$$T = \frac{263.5\,Q}{\Delta s},\quad T = \frac{263.5 \times 3600}{39.5} = 24{,}000 \text{ gpd per ft},$$

$$S = \frac{0.301\,T\,t_0}{r^2},\quad S = \frac{0.301 \times 24{,}000 \times 0.11}{(1100)^2} = 0.00066.$$

years for the system. The resultant drawdown will be the sum of the drawdown for 1000 gpm pumping for 3 years plus the added increment of drawdown for 1000 gpm pumping for 1 year. This principle of superimposing changes in pumping on the old pumping system must be understood and followed in all calculations.

To lower the water level in the shaft from altitude 5940 (December 1952 static level) to altitude 4715 (2250 level) a drawdown of 1225 ft is required. If this is to be accomplished in 2 years (730 days) it is determined from Fig. 13 that the drawdown is 116.5 ft per 1000 gpm for this period. Dividing 1225 by 116.5 determines that 10.5 times 1000, or an average of 10,500 gpm, must be pumped continuously to accomplish this lowering. Or suppose that 12,000 gpm of pumping capacity is available, it is desired to know how long it will take to unwater the shaft to the 2250 level. Because the drawdown at this rate is 12 times that shown for Fig. 13, the time needed will be that for 1225/12 or 102 ft of drawdown to take place. This is determined to be 250 days.

To determine from Fig. 14 the amount of lowering at the site of drillhole *E* for each of the above conditions, determine the drawdown for 730 days and 250 days. When multiplied by their respective ratios of pumping, values of 57.5 and 47.2 ft determined in this manner indicate that the water level will have lowered 604 (57.5x10.5) ft for the longer period or 566 (47.2x12) ft for the shorter period of pumping. It should be noted the water level at this site would be more than 621 ft above the level in the shaft, and although the 2250 level would have been recovered at the shaft, additional pumping would be necessary to unwater the ore zone sufficiently for mining.

Good mining practice might dictate several methods for obtaining additional pumping sites to lower the water table throughout the ore zone. Deep boreholes, each equipped with a pump, might be possi-

ble, but only for convenience in calculations is the following simple solution proposed. Because the 1685 level at altitude 5280 has already been partially developed, it is proposed that the level be extended into the Eldorado limestone to the site of drillhole *E* and that a vertical winze be sunk to obtain water.

As a mathematical substitute for the winze and its short radial drifts for pump stations, a well 50 ft in radius at the site of drillhole *E* is proposed to unwater the orebody. Fig. 15 shows the drawdown calculated by the Theis nonequilibrium method for such a well pumping at the rate of 1000 gpm. The computations use a transmissibility of 24,000 gpd per ft and a coefficient of storage of 0.00067, which are average values for drillholes *E* and *F* which tap the orebody, and are modified by an impermeable boundary 5500 ft away.

To make this illustrative example conform more closely to the conditions encountered in a typical mining problem, the following schedule is proposed for a new example of calculation. Assume that the levels in the shaft and the orebody are both to be lowered to the 2250 level in a 2-year period (730 days) and also that as soon as the 1685 level at the shaft is unwatered, drifting will begin to open up additional locations to obtain water near the site of drillhole *E*. Further assume that about 200 days will be required for construction work on the 1685 level to obtain additional water from the Eldorado at this point. Assume that the shaft will be pumped at an initial rate of 8000 gpm until the 1685 level has been unwatered.

Thus, in the new example, to recover the 1685 level at the shaft, drawdown of 660 ft is required. At the rate of 8000 gpm (660/8 = 82.5) this will be accomplished in 60 days. In the same period the level in the Eldorado at drillhole *E* will have lowered 269 (8x33.6) ft. In the next 200-day period (60 + 200) of construction the water level in the shaft will have declined 160 ft more or a total of 102.5x8 = 820 ft. In the same period (60 + 200) the level at drillhole *E* will have lowered 113 ft or a total of 47.8x8 = 382 ft. This water level is 278 ft above the 1685 level, so that additional water must be obtained in the Eldorado to permit construction work to proceed on the drift extension.

Two mathematical methods may be used to determine the amount of water to be withdrawn from the Eldorado during construction of the proposed drift from the Fad shaft to the site of drillhole *E*. Either a short ground-water trench can be assumed to coincide with the drift, or a well or winze source can be assumed at the site of drillhole *E*. Both methods require withdrawal of similar volumes of water for the same source and boundary conditions. Because a winze or well source is used in later computations, that type of source will be assumed here.

From Fig. 15 it is determined that if 278/79.5 × 1000 = 3500 gpm were withdrawn for the 200-day period the water level in the Eldorado limestone would coincide with the drift level. However, it is not physically possible to withdraw water for the whole period; therefore a larger amount must be withdrawn for a shorter time. Again for convenience, assume that the unwatering takes place in the last 100 days of the 200-day period; on that basis the withdrawal must be at the rate of 278/72.5 × 1000 = 3830 gpm.

Although Fig. 14 was constructed to determine the effect of pumping at the Fad shaft on the water level at the site of drillhole *E*, it can be shown mathematically that pumping at the site of drillhole *E* has an effect on the water level at the Fad shaft approximately equal to that shown in Fig. 14; thus this figure may be used interchangeably for pumping at both sites in the following calculations.

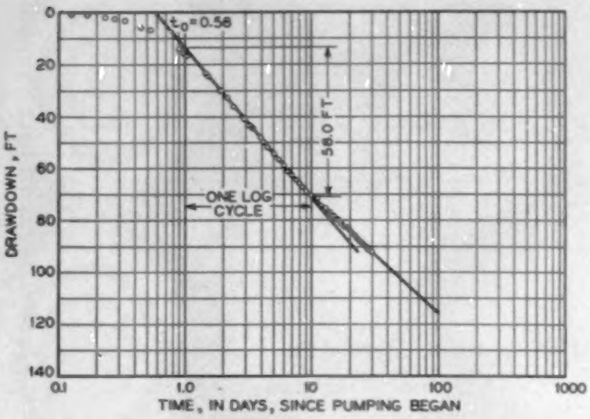

Fig. 12—A semi-logarithmic time-drawdown graph of interference of Fad shaft pumping on Locan shaft. Computations: $T = \frac{263.5\,Q}{\Delta s}$, $T = \frac{263.5 \times 3600}{58.0}$, $S = \frac{0.301\,T\,t_o}{r^2}$, $S = \frac{0.301 \times 16{,}400 \times 0.58}{(1430)^2} = 0.00014$.

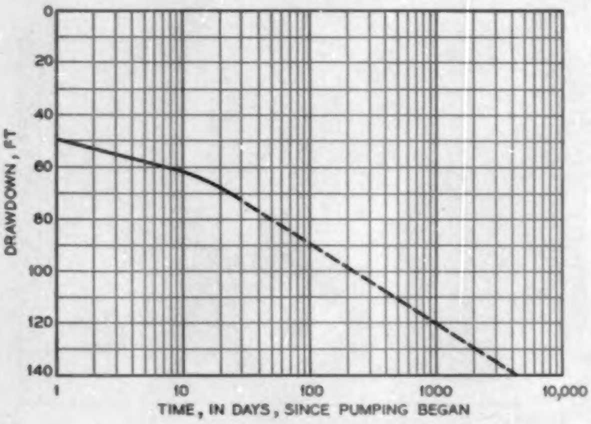

Fig. 13—A time-drawdown graph for water level in Fad shaft, pumping 1000 gpm.

Withdrawal of water from the Eldorado through the drift or winze produces an additional drawdown at the shaft of 38.5 × 3.83 = 147 ft, see Fig. 14, so that at the end of 260 days of pumping 8000 gpm from the shaft and 100 days of withdrawal of 3830 gpm from the Eldorado the water levels stand at the 1685 level in the ore block and 967 ft below the initial static level in the shaft.

In the remaining 470 days of the scheduled period the water level must be lowered 258 ft in the shaft and 565 ft in the Eldorado limestone. Continued pumping in the shaft at the rate of 8000 gpm will lower the level to (8 × 116.5 + 147) = 1079 ft. Continued pumping at the rate of 3830 gpm in the Eldorado for 470 (570 — 100) days will further increase the lowering in the shaft by 64 ft [(55.2 — 38.5) × 3.83]. Total drawdown in the shaft due to pumping at both places will be 1143 ft.

After the water level in the Eldorado has been lowered to the 1685 level additional water must be withdrawn from that formation. As stated previ-

ously, it is proposed that a winze be sunk to accomplish this withdrawal, although boreholes equipped with pumps would be equally satisfactory, and the amounts of water to be withdrawn would be about equal in each case. If the winze source is used, continued pumping in the Eldorado at the rate of 3830 gpm for 470 (570 — 100) days will lower the level 65 ft [(89.5 — 72.5) × 3.83]. Continued pumping at the shaft for 470 (730 — 260) days also lowers the level in the Eldorado by 78 ft [(57.5 — 47.8) × 8.0]. Owing to the continued pumping at past rates in both places the level in the Eldorado is 143 ft below the 1685 level or 422 ft above the 2250 level. To obtain 422 ft of lowering in the Eldorado by the end of the 2-year period the rate must be increased during the last 470 days. From Fig. 15 it is determined that 4820 (422/87.5) gpm of additional water must be withdrawn at the winze site during this period.

During this period of 470 days the increased pumping at the winze will increase the drawdown at the shaft, see Fig. 14, by 53.3 × 4.82 = 257 ft. Total drawdown at the shaft will now be 1400 ft, or 175 ft below the 2250 level. Theoretically pumping at the shaft could be reduced by 175/110.3 x 1000 = 1590 gpm during the last 470 days, so that the water level would not be below the required 2250 level. If pumping from the shaft is reduced 1590 gpm the levels in the Eldorado will rise 85 (53.3 × 1.59) ft. This would require additional pumping from the Eldorado of 85/87.5 × 1000 = 970 gpm. By means

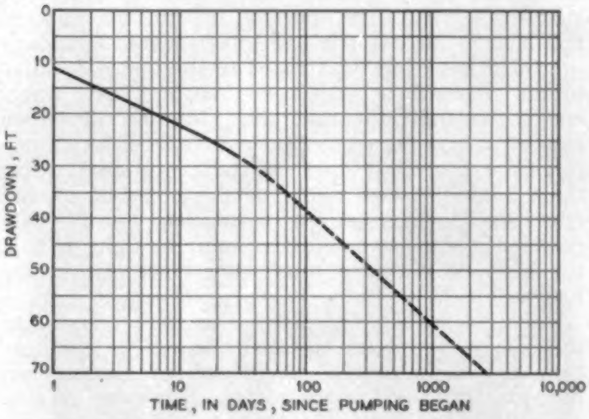

Fig. 14—A time-drawndown for water level at drillhole *E* caused by pumping 1000 gpm at Fad shaft.

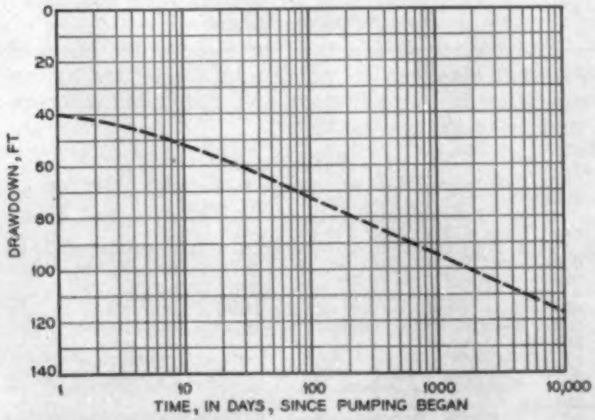

Fig. 15—A time-drawdown graph for water level in a well 50 ft in radius, in orebody at site of drillhole *E*, pumping 1000 gpm, barrier boundary at 5500 ft.

of several adjustments of this nature it is determined that about 3830 + 6200 or 10,030 gpm must be pumped from the Eldorado and about 5750 from the shaft during the last 470 days of the 2-year period.

To Summarize the Calculations: The pumping schedule indicates that the pumping rate from the shaft will be 8000 gpm for 260 days and then will gradually decrease to an average of 5750 gpm for the remainder of the 2-year period. For the Eldorado limestone there is no pumping for the first 160 days; from the 160th to the 260th day the rate will be 3830 and from the 260th day to the 730th day will average 10,030 gpm. Such a pumping schedule will lower the water level from the initial static level, altitude 5940, to the 2250 level at altitude 4715 in a 2-year period.

One detail in the complete appraisal of the water conditions remains, namely, the amount of water to be pumped to maintain the water level below the mine operations. This rate can be obtained by extension of the rates required for unwatering and reducing the rate of pumping by an amount that will allow the water level to rise an amount equal to the drawdown that would occur during the period of extension. If no boundaries were present the terminal rates of pumping could be determined by application of the formula developed by Jacob and Lohman[7] for nonsteady flow to a well in which the drawdown is constant.

Discussion of Applications of Methods and Limitations of Data: Numerical values obtained in the example were presented only to illustrate the method; values would change for any other time or rate of pumping. The actual time and rates in any appraisal are dependent on good mining practice and knowledge of the geologic conditions. However, a discussion of the hydrologic factors is needed so the mining engineer can evaluate the degree of accuracy of his calculations.

In the Eureka area the geology is complex in that the formations are discontinuous, and in places non-water-yielding formations abut more permeable formations. It might be expected that the hydrology would be similarly complex and that the infinite-aquifer type of analysis used in quantitative ground-water would not be applicable. However, the data obtained in the pumping test do not confirm any such assumption as to the complexity of the hydrology; in fact, the opposite is true. The close fit of the type curve for an infinite aquifer to the test data in Figs. 4, 6, 8, and 10 during the period of pumping indicates that hydraulic continuity exists throughout the cone of unwatering in the formations tapped by the Fad shaft, drillholes *E* and *F*, and the Locan shaft. The straightline plotting of drawdown in Figs. 5, 7, 9, 11, and 12 further demonstrates the areal hydraulic continuity; if there were discontinuous conduits in the system the plotted data would be irregular. Close correlation of test data with an infinite-aquifer analysis modified for one boundary is justification for extensions, by the Theis formula, of the drawdowns for other times, distances, and rates of pumping.

The range in the determined values for the coefficient of transmissibility, Table II, for holes *E* and *F* is small. The magnitude of the values has been checked independently by the Theim method, which uses the slope of the water table at equilibrium. This check of data collected by the Eureka Corp. Ltd. in December 1948 and February 1953 confirms the magnitude of the coefficient of transmissibility

determined by the test. The values determined at the Locan shaft are low because the shaft did not penetrate the Eldorado limestone but tapped the Ruby Hill fault zone only. The Fad shaft also does not penetrate the Eldorado. Values for transmissibility are not expected to change during the unwatering process, but if changes occur the value would be reduced.

The coefficient of storage has a very low numerical value indicative of water confined between impermeable beds. In the ore-block area the Secret Canyon shale is the confining layer, but the identity of the confining layer in the surrounding area is not known. It is probable that as unwatering progresses and the water level declines below the bottom of the Secret Canyon shale the storage coefficient will increase approximately to the specific yield of the limestone, not known but certainly larger than the storage coefficient. This would slow the rate of unwatering for the same rate of pumping.

Boundaries of the ground-water reservoir tapped by the Fad shaft are unknown at this time. If the conduits (porous formations or fault zones) extend out and under the valleys north and west of the mine the boundaries of the reservoir are many miles away. Many faults are thought to extend from Ruby Hill into the valley area, but there is no knowledge as to whether these faults may be conduits for water. Measurements of the water levels in wells 3 to 6 miles to the north and in the Holly shaft, 6600 ft north of the Fad shaft, showed no changes that could be attributed to the test pumping. However, inconclusive evidence for 1949 indicates there may have been a lowering due to pumping in 1948. The 30-day test indicates that one impermeable boundary was intersected by the cone about a mile from the Fad shaft. The data are inadequate for location of the boundary. There was no evidence of recharge from any outside source during the test.

Table III. Volume of Water Required to Unwater 1685 Level in the Fad Shaft with Pumping at 12,000 Gpm, as Compared with Volume Required with Pumping at Lesser Rates

Pumping Rate, Gpm	Days Required to Recover 1685 Level	Volume of Water Pumped in Period, Million Gallons	Ratio of Volume Pumped at Lesser Rate to Volume Pumped at 12,000 Gpm
12,000	3	51.8	1
10,000	17	245	4.73
8,000	60	691	13.3
6,000	450	3888	75.0

A resumé of all the conditions affecting the water levels and pumping rates indicates that a substantial degree of confidence may be placed in the extrapolated curves for drawdown at the Fad shaft and at the site of hole *E*. However, measurements of the rate of decline should be compared with the calculations so that ideal curves can be modified to fit natural conditions.

Rates of Pumping: Rates of pumping must be determined by judgment, modified by both the mechanical possibilities of placing the required number of pumps at a given site and the probability of obtaining these rates of flow from the formation. For example, it may not be possible to pump at high rates from individual wells or other excavations in the Eldorado, in as much as its transmissibility is not high enough. Consideration should be given also to the possible detrimental effects of pumping at high rates, for example, erosion of the fault opening on the 2250 level such as occurred in 1948 when a rate of 9000 gpm was attained in the Fad shaft.

However, rate of pumping must be high to reduce time of unwatering to a minimum to accomplish the project most economically. Assuming that cost of unwatering is directly proportional to the product of rate and duration of pumping, or the volume of water pumped, the following computations show the importance of reducing the duration or time of pumping. Table III shows the ratio of volume of water to be pumped, which is assumed to be the same as the ratio of cost, to recover the 1685 level for various rates of pumping in the Fad shaft, see also Fig. 13.

Summary: Data were collected during a 30-day pumping test at the Fad shaft, Eureka, Nev., to gain information on the water-transmitting and storage characteristics of bedrock near the shaft. Methods of analysis of the pumping-test data were those used for determining yields of aquifers to conventional well installations. The method of appraisal of the water conditions is similar to the method of determining amount and rate of lowering of the water level in a well field. To demonstrate a method where two wells (shaft and winze) are pumping from the same formation, a sample calculation is made illustrating the effects of the drawdown caused by each.

The average coefficients of transmissibility and storage for the Eldorado limestone were 24,000 gpd per ft and 0.00067, respectively. The coefficient of transmissibility for the Fad shaft was 16,400 gpd per ft. These values were used in extending drawdown curves for greater time periods, and the curves were modified to include the one impermeable boundary of which the effect was noted in the test data. Use of the curves in a sample evaluation of the water conditions indicates that the Eldorado limestone cannot be unwatered to the desired level by pumping from the shaft only, so that additional water must be withdrawn closer to the point of mining operations to unwater that formation. Sample calculations for the two points of pumping indicate that the heaviest pumping must take place within the Eldorado.

The close adherence of these particular test data to the type curves for an infinite aquifer is justification for considerable confidence in calculations based on extension of the curves. However, even in such a case the mine operator should make sufficient checks of progress when unwatering is undertaken to determine if the close correlation persists with time and to make modifications in the program.

Acknowledgment

The author is indebted to George W. Mitchell, General Manager of the Eureka Corp., for his cooperation in the collection of the detailed information pertaining to the test.

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Technical Note

Measuring the Tensile Strength of Rocks

by Rudolph G. Wuerker

THE scarcity of values of tensile strength of rocks has been explained by the lack of successful testing procedures. In the case of mine rock a description is given[1] of the difficulties encountered in testing a cylindrical specimen, such as a core, by conventional methods.

Over a period of years the following method has given definite and reproducible results with the weakest as well as with the strongest rocks. It does not completely supersede the use of cores with special fixtures but is a supplement in all cases where cores cannot be obtained, as from soft rocks, or in cases where it is less expensive to prepare test specimens by cutting them out of the rock instead of drilling cores.

Principle and equipment are the same as for the test for tensile strength of hydraulic-cement mortar.[2] The test specimen, Fig. 1, has the shape of a briquet. While in the original cement mortar test the briquet is cast in a special mold, it is prepared from rocks in different ways, depending on how easily they can be cut and shaped.

Soft rocks, which cannot be core-drilled with a carboloy or diamond bit, are simply hand-cut. Only two dimensions need be watched. The first is the 1-in. diam at the narrowest cross-section of the briquet. The other critical measure is the radius of curvature of the waistline, as the roller supports in the grips have a fixed distance. This radius is ground out of the solid by means of a carborundum grinding wheel having a ¾-in. radius.

Medium hard rocks can be core-drilled with a carboloy bit. The resulting core can be used for nondestructive sonic testing first, and after that for any destructive test. By using an EX-bit and by carefully placing the coreholes, preferably by using a tenplate such as shown in Fig. 1, it is possible to obtain from the rock a punched sample from which numerous tensile briquets can be made. The outside radius of the EX-bit differs from the radius of curvature of the briquet by ⅛ in., but this still permits placing and aligning the specimen in the grips.

In the case of bedded rocks the core might have bedding planes normal to the plane of the briquets, and rocks can be tested in any arrangement of the bedding planes desired.

Hard rocks, limestones, igneous, and metamorphic rocks can only be diamond-drilled or diamond-cut. Here the method of getting the tension briquets by accurate placing of EX-drill holes is especially economical. The tops of the briquets made from hard rocks cannot be rounded; they are straight cuts made with a diamond cut-off saw and rounded off on a polishing wheel.

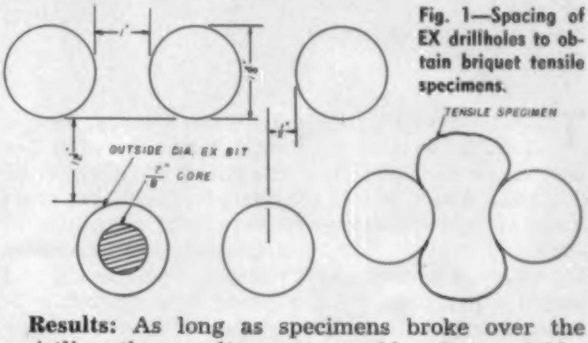

Fig. 1—Spacing of EX drillholes to obtain briquet tensile specimens.

Results: As long as specimens broke over the waistline the results were considered acceptable. Further statistical treatment of the tests[1] showed a satisfactory percentage of standard deviation.

The tensile strength values obtained by this method do not represent true values because of the stress concentration caused by the curvature of the side of the piece and because of the closeness of the grips. The ratio of maximum to average stress at the plane of failure has been determined to be about 1.75.[3] All tensile strength values listed in Table I are corrected accordingly.

To avoid this stress-concentration, if there are a sufficient number of cores, tensile strength can be measured by imbedding the cores in mortar in the two outer briquets in the gangmold.[4]

Strain-Measurements: The applicability of the briquet specimens for strain observations was tested in the case of sandstone and shale. Two element Rosette SR-4 strain gages were used. Young's modulus and Poisson's ratio, both in tension, were computed and found to be different from those in compression, determined during the same test series and from the same rock, see Table I.

R. G. WUERKER, Member AIME, is Assistant Professor of Mining Engineering, College of Engineering, University of Illinois, Urbana, Ill. Manuscript, March 12, 1954. TN 253A.

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Table I. Results of Tensile Tests

ROCK	DIORITE	SHALE	SANDSTONE	POTASH-SALT
Source	Minnesota	Roof of Seam No. 6 Bell & Zoller Mining Co., Murdock, Ill.	McCleansboro Group, Pennsylvanian, Salt-Fork River, 20 miles east of Urbana, Ill.	Mixture of halite and sylvite, Carlsbad, N. M.
Grain size	1/16 to 1/64 in.	Not recognizable	About 1/64 in.	⅛ to ½ in.
Load normal or parallel to bedding plane	Undetermined	Parallel	Normal	Undetermined
Number of specimens tested	3	10	10	2
Number of specimens giving useful data	3	7	9	2
Tensile strength, corrected value, psi, avg	2550	1538	216.5	402
Standard deviation, pct		4.7	10.7	
Young's modulus in tension, psi, avg		4.98×10^6	0.197×10^6	
Standard deviation, pct		1.52	9.14	
Poisson's ratio in tension, avg		0.432	0.142	
Standard deviation, pct		0.33	9.15	
		Compression Tests		
Young's Modulus in compression, psi, avg		1.09×10^6	0.725×10^6	
Standard deviation, pct		1.65	6.62	
Poisson's ratio in compression, pct		0.103	0.418	

Geophysical Case History

Of a Commercial Gravel Deposit

by Rollyn P. Jacobson

THE town of Pacific, in Jefferson County, Mo., is 27 miles west of St. Louis. Since the area lies entirely on the flood plain of a cutoff meander of the Meramac River, it was considered a likely environment for accumulation of commercial quantities of sand and gravel. Excellent transportation facilities are afforded by two major railways to St. Louis, and ample water supply for washing and separation is assured by the proximity of the river. As a large washing and separation plant was planned, the property was evaluated in detail to justify the high initial expenditure. An intensive testing program using both geophysical and drilling methods was designed and carried out.

The prospect was surveyed topographically and a 200-ft grid staked on which electrical resistivity depth profiles were observed at 130 points. The Wenner 4-electrode configuration and earth resistivity apparatus* were used. In all but a few cases, the electrode spacing, A, was increased in increments of 1½ ft to a spread of 30 ft and in increments of 3 ft thereafter.

* Manufactured by the Geophysical Instrument Co., Division of the Generator Corp. of Manassas, Va.

Initial drilling was done with a rig designated as the California Earth Boring Machine, which uses a bucket-shaped bit and produces a hole 3 ft in diam. Because of excessive water conditions and lack of consolidation in the gravel there was considerable loss of hole with this type of equipment. A standard churn drill was employed, therefore, to penetrate to bedrock. Eighteen bucket-drill holes and eight churn-drill holes were drilled at widely scattered locations on the grill.

The depth to bedrock and the configuration will not be discussed, as this parameter is not the primary concern. Thickness of overburden overlying the gravel beds or lenses became the important economic criterion of the prospect.**

** The term *overburden* is used to signify all material overlying gravel or sandy gravel. *Soil* and *mantle* refer to surface soil and secondary material derived from recent flooding.

The wide variety and gradational character of the geologic conditions prevailing in this area are illustrated by sample sections on Fig. 2. Depth profiles at stations E-3 and J-7 are very similar in shape and numerical range, but as shown by drilling, they are measures of very different geologic sequences. At J-7 the gravel is overlain by 15 ft of overburden, but at E-3 bedrock is overlain by about 5 ft of soil and mantle.

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Discussion on this paper, TP 3872L, may be sent (2 copies) to AIME before April 30, 1955. Manuscript, Feb. 16, 1954. New York Meeting, February 1954.

Stations L-8 and H-18 are representative of areas where gravel lies within 10 ft of surface. In most profiles of this type it was very difficult to locate the resistivity breaks denoting the overburden-gravel interface. In a number of cases, as shown by stations M-4 and H-18, the anomaly produced by the water table or the moisture line often obscured the anomaly due to gravel or was mistaken for it. In any case, the precise determination of depth to gravel was prevented by the gradual transition from sand to sandy gravel to gravel. In spite of these difficulties, errors involved in the interpretation were not greatly out of order. However, results indicated that the prospect was very nearly marginal from an economic point of view, and to justify expenditures for plant facilities a more precise evaluation was undertaken.

The most favorable sections of the property were tested with hand augers. The original grid was followed. In all, 46 hand auger holes were drilled to gravel or refusal and the results made available to the writer for further analysis and interpretation.

When data for this survey was studied, it immediately became apparent that a very definite correlation existed between the numerical value of the apparent resistivity at some constant depth and the thickness of the overburden. Such a correlation is seldom regarded in interpretation in more than a very qualitative way, except in the various theoretical methods developed by Hummel, Tagg (Ref. 1, pp. 136-139), Roman (Ref. 2, pp. 6-12), Rosenzweig (Ref. 3, pp. 408-417), and Wilcox (Ref. 4, pp. 36-46).

Various statistical procedures were used to place this relationship on a quantitative basis. The large amount of drilling information available made such an approach feasible.

The thickness of overburden was plotted against the apparent resistivity at a constant depth less than the depth of bedrock for the 65 stations where drilling information was available. A curve of *best fit* was drawn through these points and the equation of the curve determined. For this relationship the curve was found to be of the form

$$\rho = b\,D^{n}$$

where ρ is the apparent resistivity, D the thickness of overburden, and b a constant. The equation is of the power type and plots as a straight line on log-log paper.

The statistical validity of this equation was analyzed by computation of a parameter called Pearson's correlation coefficient for several different depths of measurements, see Ref. 5, pp. 196-241. In all but those measurements taken at relatively shallow depths, the correlation as given by this general equation was found to have a high order of validity on the basis of statistical theory.

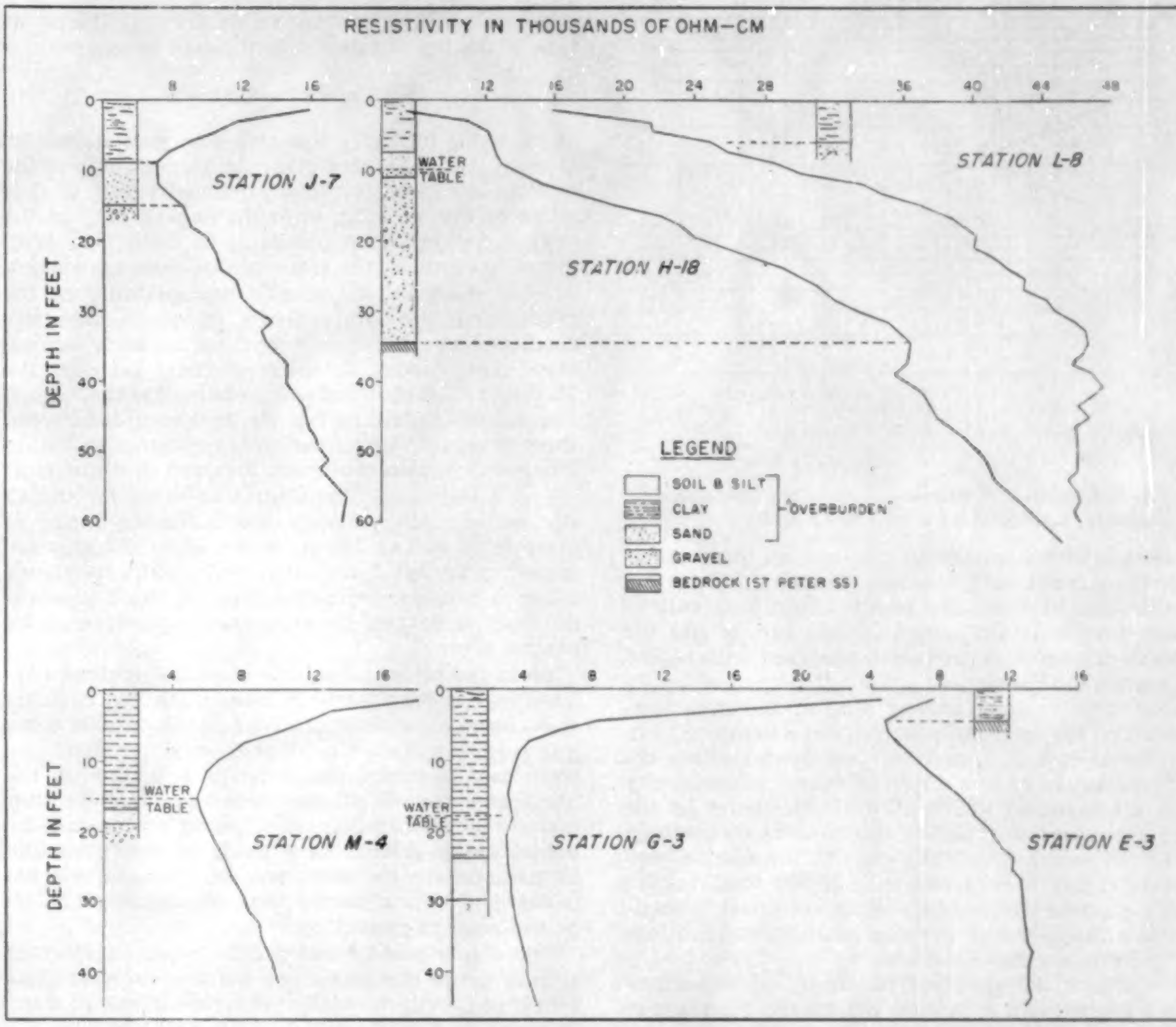
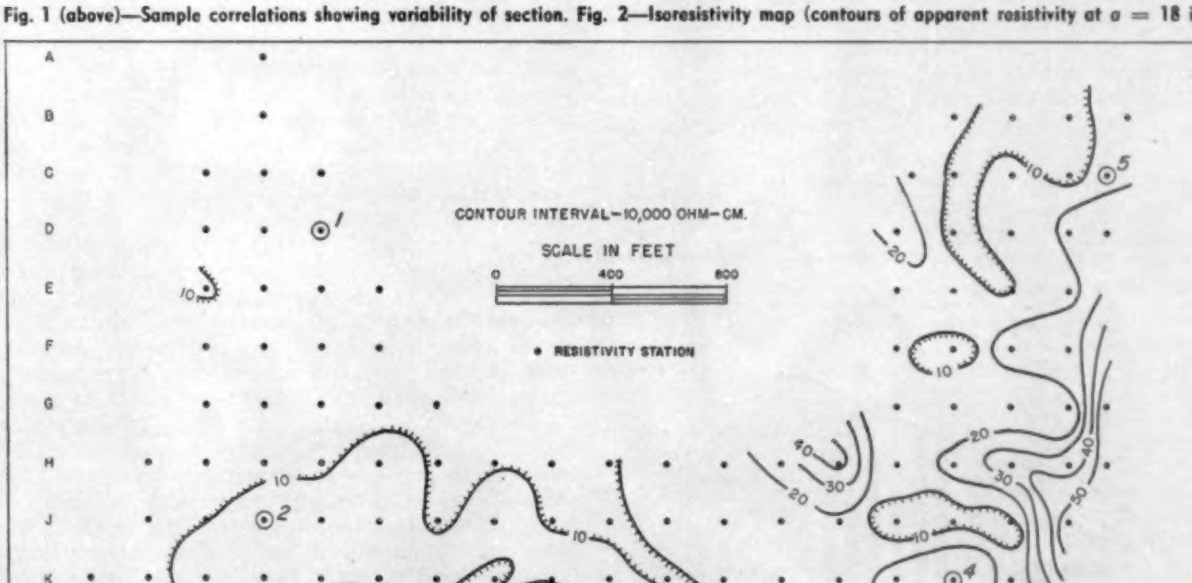


Fig. 1 (above)—Sample correlations showing variability of section. Fig. 2—Isoresistivity map (contours of apparent resistivity at $a = 18$ in.)



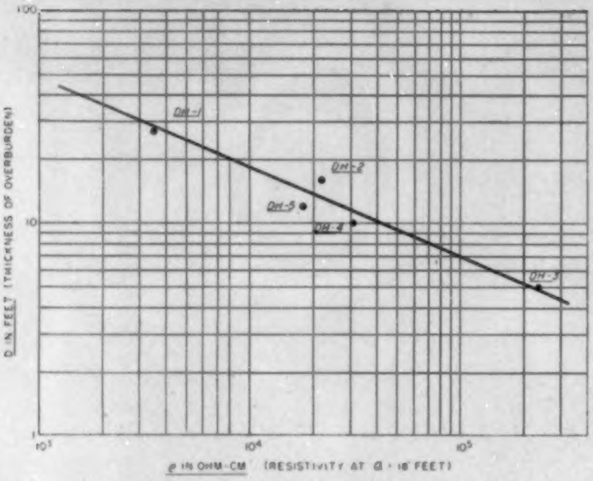


Fig. 3—Correlation of thickness of overburden with apparent resistivity. Equation of curve: log 6.85-2.2 log D.

Details of the arithmetic involved in these calculations will not be presented. Rather than this, an application of the basic principle outlined will be made to the data obtained in this survey and the results of empirical prediction analyzed with regard to known conditions.

Fig. 2 is an isoresistivity map of the area contoured on the apparent resistivity at a depth of 18 ft. Drillholes 1, 2, 3, 4, and 5 (large open circles) are representative of five different ranges in resistivity and are assumed to reflect the dependency of the apparent resistivity on the thickness of overburden over the entire area. For ease of illustration, apparent resistivities greater than 50,000 ohm-cms are not contoured. (Maximum values are given in thousands of ohm-cm by the small numbers at stations where this maximum occurs.)

On Fig. 3 the apparent resistivity, ρ, at each of these five stations is plotted against the thickness of overburden, D, as determined by drilling. The equation of the best straight line through these points is

$$\log \rho = 6.85 - 2.2 \log D$$

($6.85 = \log b$). This equation was then solved for D at all stations on the grid. An isopach map of the overburden based on these predicted values of D is given in Fig. 4a. Fig. 4b is an isopach map of the overburden based on results of 65 drillholes. With minor exceptions, the maps are in close agreement.

As a check on the overall dependability of the predictions, the total volume of overburden was computed by planimetric methods on each isopach map. The volume of material lying between the 10-ft and 20-ft contours was measured, as the isopach map based on drilling has the best control between these contours. Computed volumes agree to within 1 pct, well within the errors involved in contouring.

Fig. 5 shows the distribution of error for the 65 stations drilled. Although the maximum range of error is 14 ft, two-thirds of the 65 predictions are shown to be less than 2 ft in error. The frequency curve is roughly symmetrical about the 0-value of no error, indicating the compensating nature of the overall error.

As in the present case, the empirical method outlined here is very useful in integrating and evaluating a large number of depth profiles as a unit using a few representative drillholes; only five drillholes were used to derive the equation as applied in the previous topic. Predicted thicknesses (or depths) can be related to changes in slopes or *breaks* on individual depth profiles as a guide to interpretation. In many cases very minor anomalies can be selected in favor of strong breaks that are unlikely choices on the basis of prediction.

There is a second and possibly more important application of this procedure conjunction with horizontal resistivity profiling. The general use of hori-

Fig. 4a—Isopach map of overburden based on empirical results.

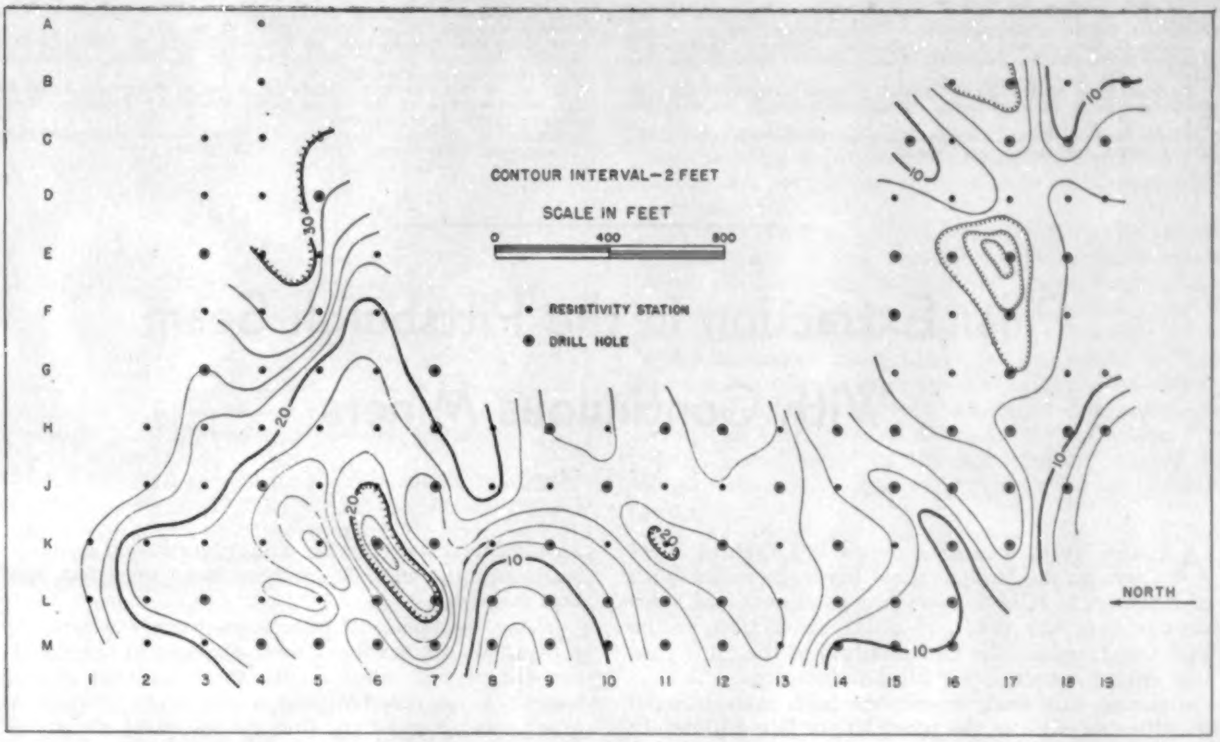


Fig. 4b—Isopach map of overburden based on drilling results.

zontal profiling has been as a rapid reconnaissance tool to evaluate geological conditions qualitatively. Ordinarily very little information of a quantitative nature is derived from such surveys without resort to costly drilling and depth profiling. The following procedure outlines the merits of a rapid, economical, and reliable method of realizing the quantification of horizontal profiling. 1—Construct an isoresistivity map of the field data. 2—Drill a minimum number of points representative of the total range in apparent resistivity. Obviously, these drillholes must be located at points where electrical measurements were taken. 3—Derive an empirical equation relating the apparent resistivity at some constant electrode spacing to the geologic condition of concern, i.e., thickness of overburden, depth to bedrock, and thickness of some intermediate horizon. 4—Solve this equation for all points where electrical measurements were taken. 5—If check drilling is performed, at least as many holes must be drilled as were originally used to define the empirical equation used. This necessity is apparent in view of the nature of the distribution of error as shown in Fig. 5. On the basis of probability, one out of four holes would show an error greater than 2 ft and improper evaluation of the method would result if only one such hole were drilled.

From an economic point of view, this method does not involve the application or expense of any new technique or equipment not already a part of standard electrical resistivity surveying. The drilling information required is not in excess of the usual requirements for adequate calibration of electrical data. In fact, necessary drilling might easily be reduced by the method. It must be emphasized that if no correlation between variables exists (a basic necessity of the method), little or nothing has been lost in the way of time or expense. The basic field procedures remain the same and the resultant data can be interpreted by ordinary methods.

Summary and Conclusions

A very rapid and simple empirical method of interpretation of electrical resistivity data has been presented. The method has shown itself to be very useful and reliable in application to a practical problem. Applications of the method to other problems of the same general nature have been suggested and general procedures outlined. If the basic assumption necessary to the method proposed does not prevail in a given problem, all other interpretive measures can still be realized, as no change in obtaining the basic field data are required.

The author expresses thanks to the Missouri-Illinois Materials Co., St. Louis, Mo., for permission to publish this paper.

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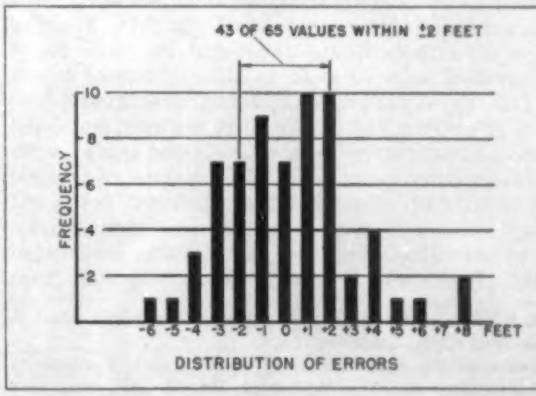

Fig. 5—Distribution of errors in prediction.

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Pillar Extraction in the Pittsburgh Seam With Continuous Miners

by W. E. Hess

AT the Vesta mines of Jones & Laughlin Steel Corp. on the Monongahela River, 35 miles south of Pittsburgh, JCM Joy continuous miners and 6-SC shuttle cars are used for pillar extraction in the Pittsburgh seam. The entire output of the mine goes into coke production for blast furnace operation.

Because this coal, which has high metallurgical qualities, is close to the place where it is ultimately used and because supplies with these properties are being rapidly depleted, complete recovery is essential wherever possible. With the introduction of the continuous miner into the Vesta mines, therefore, it was necessary to plan a project which would facilitate development and provide for future total recovery. Use of continuous miners under varying conditions has necessitated changes with regard to roof action, method of development, method of pillar attack, and innovations in mining. In the following pages these changes will be discussed.

Prior to the introduction of continuous miners the Pittsburgh seam at the Vesta mines averaged 72 in. of mineable coal, overlain with 12 to 16 in. of drawslate, which weathers on exposure to air and is not firm enough to be held in place. Immediately overlying the drawslate is a wild or *rooster* coal, very irregular in thickness, varying from 12 to 30 in. Above the rooster coal the strata consists predominately of shales or lime shales and sandstone. Just before the introduction of continuous miners in the Vesta mines, however, a definite change took place in the character of the roof structure overlying the coal seam, see Fig. 1.

Thickness of laminated roof immediately overlying the rooster coal varies with localities and introduces a problem in roof control. In fact, a set of eight main entries being developed in Vesta No. 4 mine entered an area with so difficult a roof structure that management considered abandoning the section altogether. In this locality the roof consisted of thin alternate layers of roof coals and shales, with an occasional irregular seam of slick slate, the whole being laced with slips, and the coal itself being cut by clay veins and spars. In an effort to overcome some of the difficulties, roof bolting was introduced in 1949. This made it possible to recover coal from an area which otherwise would have been economically unmineable. There have been very few roof bolt failures.

When conventional machines were replaced by continuous miners there were changes in the development of butt entries. By these changes it was hoped: 1—to speed development, 2—to provide as much solid coal as possible for the machine on retreat, 3—to avoid the necessity of crossing entries already driven, 4—to facilitate ventilation, 5—to provide a roadway for shuttle car haulage which would eliminate the necessity of running through doors or curtains, 6—to provide better roof control by increasing the distance between the center lines of entries, and 7—to provide more coal when the machine was in position to load.

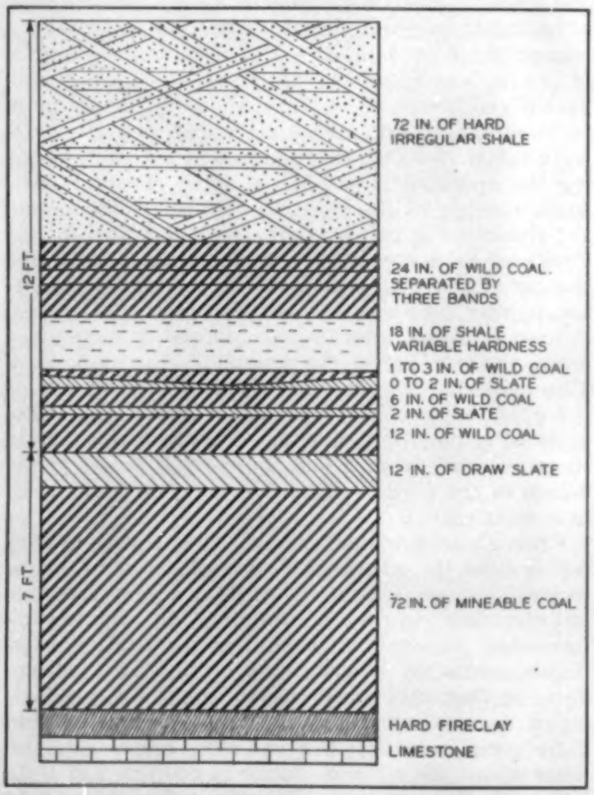

Fig. 1—Stratigraphic column of present coal-producing area. Thickness of laminated roof overlying the rooster coal varies, introducing roof control problems.

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Discussion on this paper, TP 3910F, may be sent (2 copies) to AIME before April 30, 1955. Manuscript, Jan. 13, 1954. New York Meeting, February 1954.

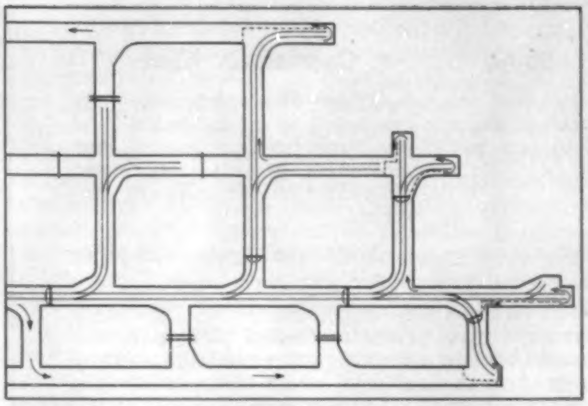

Fig. 2—The present method of development for pillar extraction in the Pittsburgh seam with continuous miners.

The ventilation plan for working faces in a three-entry development butt entry placed the middle entry on the intake, the volume of air dividing right and left and returning on the outside entries. This method makes it possible to eliminate all doors and curtains on the haulways and assures that all dust created by the machine operation is carried away from the operator and is returned behind the line brattice.

Roof control on development is important so that entries will remain open for retreat work without an excessive amount of retimbering. Roof bolting can be of material help but must be done in conjunction with the operating cycle. Although the greatest benefits are obtained from roof bolting performed immediately after coal extraction, more men are required than for conventional timbering, and there are longer periods of delay. For these reasons roof bolting was not employed in development with continuous miners.

Since the drawslate overlying the Pittsburgh seam weathers on exposure to air and is not firm enough to be held in place, it must be taken by the continuous miner as the entries are developed, and more maintenance is required than is needed on machines extracting coal only. Since the machine tears loose large flakes as it cuts through the coal and slate, it does not have the grinding action that may be anticipated with the usual cutting. This is due in part to the fact that the slate is open on three sides and separates easily from the coal roof immediately overlying it.

Several factors influenced the decision to carry out all development with conventional machines so that the continuous miners could be taken off entry development and placed on retreat work on pillaring: 1—the severe treatment of the continuous miners during cutting of drawslate in solid work; 2—the possibility of setting roof bolts for better roof control wherever conventional machines are used; 3—the introduction of hand-held hydraulic drills mounted on the cutting machines, which made it possible to eliminate one working place and use a smaller crew; 4—the need for a more economical method of providing loading points at a right angle to the mine car, which would require a minimum time for preparation; and 5—the fact that coal produced by the continuous miner on retreat is much more coarse and the slate in larger pieces than that produced by the same machine in development. This is a considerable help in the preparation plant.

Fig. 2 shows the method of development used for pillar extraction in the Pittsburgh seam with continuous miners. The number of entries has been increased from three to four, principally to supply sufficient working places for all crew members to work. The block of coal between the haulway and the solid rib is smaller in size, as it gets some protection from the solid coal and permits rapid extraction. Loading points are established at the mouth of each room, permitting entry stumps to be extracted in regular sequence.

Since there was better success with rooms driven on the butts, it was decided to do as much room work on the butts as possible. Accordingly, two rooms were driven on the faces, and from these it was planned to drive rooms on 25-ft centers, 13 ft wide, leaving a 12-ft pillar, which would be recovered on retreat after the rooms had been driven a distance of 200 ft. The character of the roof overlying the seam would not permit continuing this method of mining, as the small pillar started to crush, leaving no proper protection for equipment and men.

In the next plan, rooms were driven on 100-ft centers, one at a time. After 100 ft a breakthrough was turned and driven into the gob area. The machine dropped back, halfway through the breakthrough, and drove a place through to the gob on the same course as that on which the room was

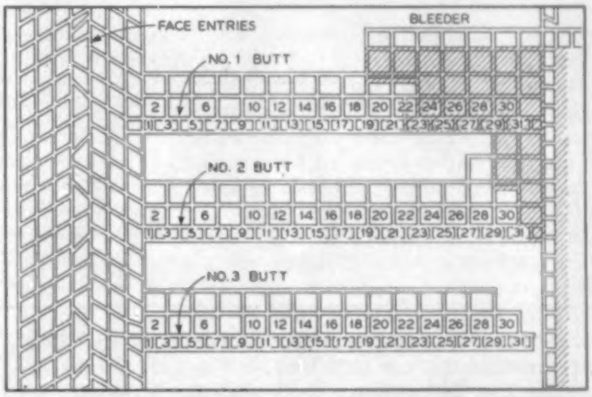

Fig. 3—The sequence of driving the individual rooms and pillar lifts according to the present method is indicated by consecutive numbers.

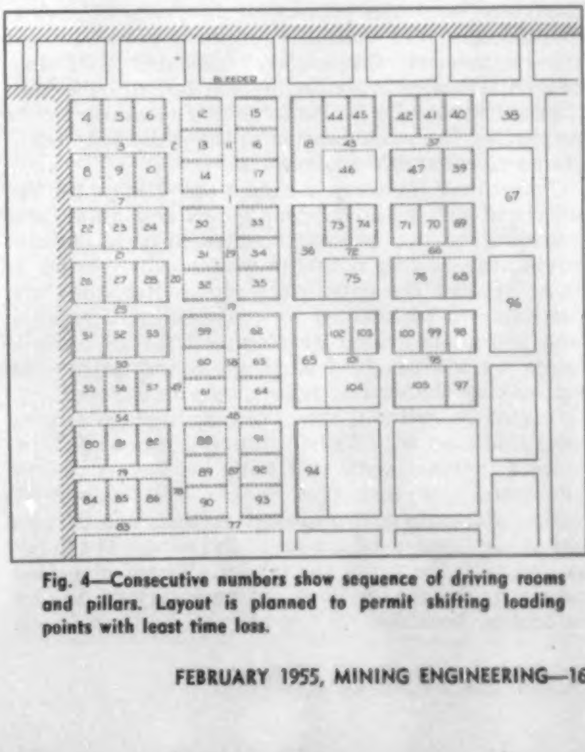

Fig. 4—Consecutive numbers show sequence of driving rooms and pillars. Layout is planned to permit shifting loading points with least time loss.

Hand Loading	Mechanical Loading	Continuous Miner
1—Long pillar drive, extending over several butt headings, to obtain production.	1—Shortening of pillar line, averaging 8 to 12 working places, generally confined to one butt heading.	1—Pillar line shortened. Only one working place available at one time for best control, less movement of equipment, and possibility of less operating personnel.
2—Slow progress of the pillar line.	2—More rapid progress of pillar line.	2—Very rapid progress of pillar line.
3—Inability to control because of unequal ability of personnel, absenteeism, presence of natural difficulties (spars, etc.).	3—Exclusive control by foreman in charge, who designates where equipment will work and assigns working positions to men to encourage leading in production on multiple-shift operation.	3—Impossibility of deviation from controlled plan; only one working place available, and any change would throw pillar line out of sequence.
4—Absence, of necessity, of very closely controlled ventilation, time being a factor in release and dissipation of methane.	4—Controlled ventilation necessary; however, if any one place is in trouble, others are available.	4—Urgent necessity of controlled ventilation, as the one working must be kept free from place accumulation of gas caused by generation, roof falls, etc.
5—Absence of large quantities of dust in suspension.	5—Dust problem present, but not of prime importance.	5—Dust problem very acute, requiring close attention.
6—Less rigid timbering requirements.	6—Timber requirements more rigid.	6—Very rigid timbering requirements, as machine is working very close to gob area, and protection and warning to operator must be provided.
7—Men at working places idle until others finished.	7—Men at working places remain idle only briefly while others on same pillar line work.	7—Since machine works in one place, pillar line is kept under control, although one shift may suffer from lack of production.
8—No necessity for rock dusting.	8—Possibility of rock-dusting on an off shift by personnel other than crew members.	8—To comply with present requirements, rock dusting must be done on shift by operating personnel.

driven. From this *pillar split,* three 15-ft lifts were driven through the half block. The 8-ft fender between the lift and the gob was recovered as the machine retreated and afforded protection for equipment and men until the coal was totally extracted. Driving through to the gob then completed the original room, and the process of lifts was repeated. Two rooms were completed before entry stumps were recovered. Experience indicated that this method left stumps extending too far into the gob, required too long a period to prepare loading points for moving from one location to the next, and necessitated considerable track work.

Consecutive numbers in Figs. 3 and 4 show the sequence of driving individual rooms and pillar lifts according to the present method, which permits moving the loading points in a very short period of time. Because the entries are roof-bolted, they are less liable to be affected by roof failure. Preparations are made during development for all loading points, by widening out when the rooms are turned and making the loading point cavity in the roof.

Originally, without roof bolting, haulage entries were timbered with 7x9-in. timbers; use of 5x7-in. timbers together with roof bolts has helped to reduce costs.

All pillar lifts are timbered as they are driven, and at least two cribs are set at the mouth of the lift on line with the rib of the fender. When retreating the rib or the fender, it is not necessary to do any timbering, because the operator at no time is out from under the timber set during the time that the pillar lift is driven.

Since the continuous miner is confined to one working place, and it is imperative that the machine not be held up because a previous lift has not fallen or caved, particular attention must be given to the recovery of all coal in the fender which would tend to retard the fall. If for any reason the fall does not come on schedule, it has a tendency to increase the pressure on the fender and so weakens the roof over the lift being mined.

As stated previously, the coal and slate produced by the continuous miner on retreat is a coarser product than that produced by the same machine on development. The main factor affecting this is that the machine on development must cut to the top of the seam and through the drawslate. When pillars are extracted it is seldom necessary for the machine to cut more than two-thirds of the way up the seam; the remainder has a tendency to fall and then can be loaded up. For this reason most of the slate remains in good-sized pieces and in this condition is less harmful to the water system in the preparation plant. When extracting pillars, it is not necessary to use as much water in the spray system as on development, because it is not usually necessary to cut the drawslate. Cutting hard drawslate produces a large amount of fine dust which is thrown in suspension. Drier coal can help materially in the preparation plant, especially when dry screening is employed ahead of any washing.

As stated earlier in this discussion, it is imperative in the mining of the Pittsburgh seam that the largest percentage of recovery possible be achieved. In operating the miners on complete pillar extraction experience has shown that recovery has been improved 14 pct as compared with that obtained by conventional mechanical units, despite the fact that the natural conditions under which the continuous miners are working are far less favorable than conditions under which the conventional units have operated. This is especially true when their total recovery is compared to that obtained by hand loading. To date extraction of pillars on 8 butt entries has been completed, with a very small loss of coal in pillars due to any cause other than natural conditions, such as spars or clay veins. Because of roof conditions and because no territory or entry can be allowed to stand, all blocks that have been developed are now extracted and bleeders are provided at the top end of the butt headings.

Extracting pillars with the continuous miners requires more attention to detail and the following of a more definite plan for extraction than any method heretofore employed. Just as the entire personnel of a mine, supervisor and worker alike, had to change their line of thinking when changing from hand-loading to mechanical loading, operating personnel will have to be re-educated to cope with the new principles of concentration and roof control accompanying this modern method of pillar extraction. Some of these details are enumerated here for comparison, see opposite page.

The history of the mining industry records innumerable periods of shutdowns from one cause or another, some of short duration, others long. These have been due to local and national strikes, economic conditions, and planned economy. Each suspension of operations presents a problem to operating personnel, as it entails the protection of working places underground, especially on pillar lines when the suspension of operations cannot be planned in an orderly manner. These suspensions of operations have been a major contributing factor to the loss of coal in pillars which have been subject to undue pressure during the idle period. The loss of coal is more serious in conventional mechanical loading and hand-loading sections than on continuous miner pillar lines, where there is only one active working place. It is equally true that the re-opening of any pillar line after a suspension of operations costs an exorbitant amount of money, because of the number of working places that must be made available for the full complement of the crew, while the continuous miner requires that only one place be made available.

The power demand for continuous miners extracting pillars is by no means as great as it is for machines on development. Machines on development must rip the coal from the solid face, and while this is severe, it does not equal the work load placed on the machine when the drawslate must be cut. On development, power demands usually go up to 100 kw, the extreme demand being on the initial sump and at the time drawslate is cut; power demand on pillar work usually is 65 to 70 kw.

It follows that continuous miners on development in the Pittsburgh seam require considerably more maintenance than those on pillar extraction because of the severe treatment given the machine and the work load imposed on it. The cutter head, in its ripping action, goes from cutting coal to cutting very hard drawslate, and the attendant shock of the loader cutting from coal to slate is transmitted from the cutter chain back through all the gear train of the machine. It also has an effect on the cutter chains, messenger bearings, and frames of the machines. Bit costs of approximately $0.16 per ton on development have been high, mainly because of chipping of the tungsten carbide bits, while machines on retreat have bit costs of approximately $0.06 per ton.

Since the continuous miner is working in only one place at a time, it would seem that the foreman could supervise more than one machine. However, so much attention must be paid to the detail of each operation that a foreman for each crew more than pays his way in increased production per unit shift, coal recovery, and protection of equipment and working places. Rooms and pillar lifts must be turned at the proper time and place, each lift must be driven the correct width to leave sufficient protection against the gob to insure recovery of all stumps so that no weight will be carried over into the active works, particular attention must be paid to the timbering and cribbing of each place, rock dusting must be done at the proper time, and advantage must be taken of any minor delays so that timbering or rock dusting can be done.

Although at present all the continuous miners are being operated on pillar recovery in the Pittsburgh seam, it is planned to do some developing with the miners in the future. Present economics point to continuance of the adopted practice, since it is believed that an attempt to provide enough working places for a conventional mechanical loading section in the territory now being worked would offer a severe handicap.

Technical Note

Pillaring with Continuous Miners

by Stephen Krickovic

As it is commonly understood in the bituminous coal mining industry, pillaring means removal, as completely as is practical, of all pillars formed in the development of headings and rooms on *first* mining. The degree to which this can be accomplished, termed *percent of recovery*, is of course dependent generally upon the physical conditions of the bottom and the overburden, the system of min-

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TN 235F. Manuscript, March 2, 1954. New York Meeting, February 1954.

ing used and the strictness of enforcing this system; the continuity of operation; and the success of making the first major roof break on a new pillar line. While these factors presented a challenge and at times insurmountable problems, even to good miners in hand loading operations both past and present, their effect on the cost of mining was usually not too serious because the distress tonnages were relatively small.

But this is not the case with mechanical mining today. A disturbance in one or two working places on a pillar line can upset production and the cost from that particular section seriously. In one mine the total daily production was reduced by about 10 pct during a limited period because of distress in two of the six active pillar sections. Needless to say, the cost suffered comparably.

Undoubtedly, there are many who have experienced pillaring problems with conventional loaders under top that was difficult to support. Some may have decided that pillar mining was impractical under such conditions. Also, they may know of examples of troublesome pillaring with top that is considered average, and it is this situation particularly which points to the damage done to roofs by conventional mining. Blasting shatters the immediate roof in varying degrees, producing a dead weight of broken rock that often defies practical or economical timbering. With its aggravation of the adverse effect produced on the immediate roof by the major weight along a pillar line, this can be largely responsible for higher pillaring costs resulting from close timbering and appreciable retimbering, from cleanup of falls, and from lack of adequate space for free maneuverability of equipment. There are many examples of performance by conventional mobile loaders in development that are superior to those in pillars in the same mine by approximately 25 pct of the total face labor cost.

Is it possible for continuous miners to alter this picture? The answer is yes. As to how, that will depend on conditions in each mine. First of all, it is necessary to recognize a few basic changes in the usual concepts of roof control as a result of continuous mining. This can best be illustrated by several actual examples.

In a northern West Virginia mine, in which the Pittsburgh seam is mined, the author observed on several occasions the conditions in a room heading which was developed by a Joy continuous miner and retreated under a plan of full extraction. Despite the fact that very little timbering (bolts) was used in development, no falls had occurred in the two headings and their breakthroughs adjacent to the mined-out area for a distance of approximately 600 ft. In fact, no falls were found even in some of the breakthroughs ending in the gob and without the customary line of breaker posts. Top conditions along the entire pillar line could be described as excellent. In a comparable section with conventional loading facilities the top conditions were appreciably inferior.

Similarly, the use of the Marietta boring type of continuous miners was observed in an Illinois mine both in development and in retreat (pillaring). Here, as in the first example, only relatively few bolts were used in development. However, the timbers used in the West Virginia mine were not used here in pillar lifts and in retreat of fenders. If during mining in a lift the roof became heavy, the miner was backed out and the roof bolted. If that did not suffice, the miner was moved out and the remaining block of coal, if appreciable in size, approached from another position. Actually, the roof support in the mining of a standard pillar was obtained by leaving small stumps of coal which were crushed out by the 800 ft of cover. This crushing was facilitated somewhat by a soft streak in the seam. Lack of disturbance by blasting of the immediate roof, the arch effect from boring, and the speed of retreating a whole pillar are largely responsible for the successful pillaring operation.

Another example of roof behavior under continuous mining can be found in the operation of a German planer in a southern West Virginia mine in which the Pocahontas No. 4 seam is mined. Here the seam is overlain directly with about 30 ft of laminated shales and slates. Additional shales and slates, together with sandy shales and sandstones ranging in bed thickness from 15 to 80 ft, comprise the 500 to 900 ft of cover. Considerable roof pressure is evident in the conventionally mined development areas. This is marked by very heavy top and by the finely laminated beds of slates or shales appearing directly above the coal seam after the roof falls out or after it is shot for haulway clearance.

On the semi-longwall face the picture is changed. Despite the fact that roof pressure is exerted, and sometimes appreciably, on the steel props and beams and wood cribs, the fine laminations are nonexistent. The author has observed a number of times a mass of solid rock 4 to 6 ft thick and 10 to 20 ft long which had fallen in the normal breakage of the roof behind the back line of supports. While this is a different type of continuous mining, it nonetheless illustrates the change in character of the immediate roof.

Thus the behavior of the immediate roof, which is usually the troublesome portion, is favorably different in continuous mining from that in conventional mining. The time element and the retention of practically the original character of the roof will facilitate more successful methods of pillaring. Mining men will need to orient their sights, therefore, on how best to handle the roof. Old ideas may be totally outmoded. What is considered questionable and dangerous under present methods may prove to be the opposite in the coming mechanized mining era.

The final point about pillaring with continuous miners pertains to the increased productivity of the units. It has been observed in some cases that the rate of performance in pillars is twice that in development. Assistance from the roof weight undoubtedly plays a major part in this very large difference. It may be concluded, therefore, that continuous mining plans should include pillaring, unless the economic advantage would be nullified by physical characteristics of the seam and the roof which would prevent uniform recovery. It should be noted here also that the assistance from roof weight can be obtained to a certain overall degree in partial mining. In such plans more satisfactory recovery can be achieved than is being accomplished today by conventional mining, primarily because there are fewer working places required and consequently speed of mining is increased.

It should be emphasized that continuous mining involves a new and different concept. Although there will be drawbacks, its potential, intelligently exploited, is tremendous.

Structural and Stratigraphic Control of Ore Deposition in the West Shasta Copper-Zinc District, California

by A. R. Kinkel, Jr.

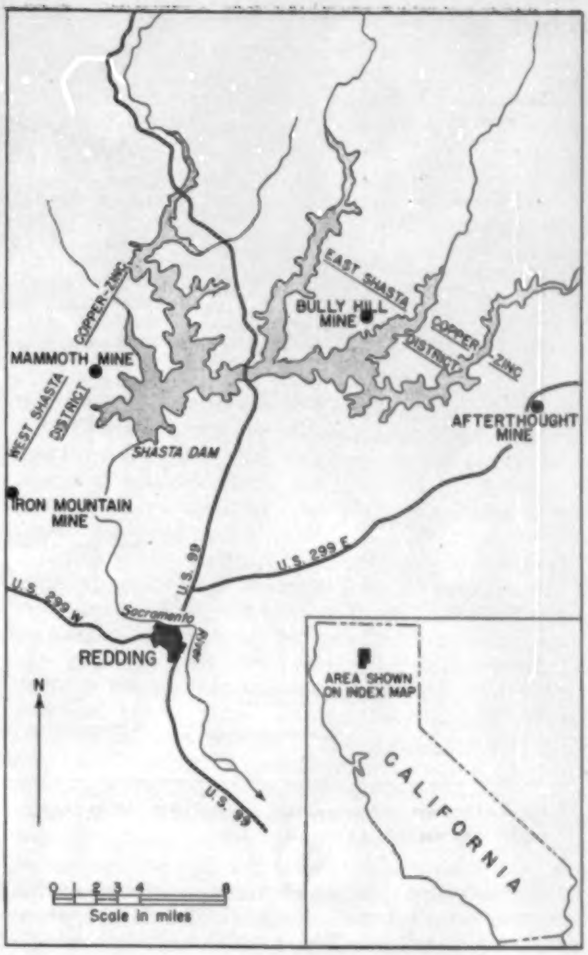

Fig. 1—Index map of the Shasta copper-zinc district, California

THE Shasta copper-zinc district of northern California lies in the foothills of the Klamath Mountains at the northern end of the Sacramento Valley. It contains two main areas of base-metal ore deposits, the East Shasta and West Shasta districts shown in Fig. 1. The East Shasta includes the area from the Afterthought mine, near the settlement of Ingot on U. S. highway 299 East, to the Bully Hill mine, on the north side of the Pit River, 9 miles to the northwest. The West Shasta, a well-defined northeast-trending district about 8 miles long and 2 miles wide, west of the Sacramento River, Fig. 2, is the western part of the ore-bearing area formerly known as the Shasta copper belt. Referred to in the older writings as a *copper arc*, this belt was thought to form a crescent extending entirely around the head of the Sacramento Valley, the convex side to the north. Recent studies by the U. S. Geological Survey in cooperation with the California State Division of Mines show that there are two districts, one at each end of the so-called copper arc, which are distinct in geologic structure and ore occurrences, and that there is only sporadic and unconnected copper mineralization between them.

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Discussion on this paper, TP 3966I, may be sent (2 copies) to AIME before April 30, 1955. Manuscript, Aug. 20, 1954. Chicago Meeting, February 1955.

Ore deposits of the two districts are similar in mineralogy, that is, they are largely massive pyritic ores containing copper and zinc with minor amounts of gold and silver, but the structure and structural control of ore deposition are different. The ores of the East Shasta district are replacements of steeply dipping shear zones in Triassic rhyolite and shale, whereas those of the West Shasta district are replacements of gently dipping rhyolite of Middle Devonian age and are essentially flat-lying. Thus there is marked lateral control of orebodies by shear zones in the East Shasta district and equally marked vertical control of orebodies by the stratigraphy in the West Shasta district.

Geologic Setting

The Paleozoic rocks of the West Shasta district range in age from Middle Devonian (?) to Mississippian. The oldest formation exposed is the Copley greenstone, probably Middle Devonian. It is composed of volcanic flows, volcanic breccias, and tuffs, all of mafic composition, but contains minor amounts of shale and rhyolitic tuff. Largely submarine in origin, it rests upon a base of older marine sediments, as shown diagrammatically in Fig. 3. The lower part of the formation is made up principally of flows, many of which are ellipsoidal, and the upper part contains most of the pyroclastic material. The formation is at least 3700 ft thick.

The Balaklala rhyolite overlying the Copley greenstone is made up of many rhyolitic flows and beds

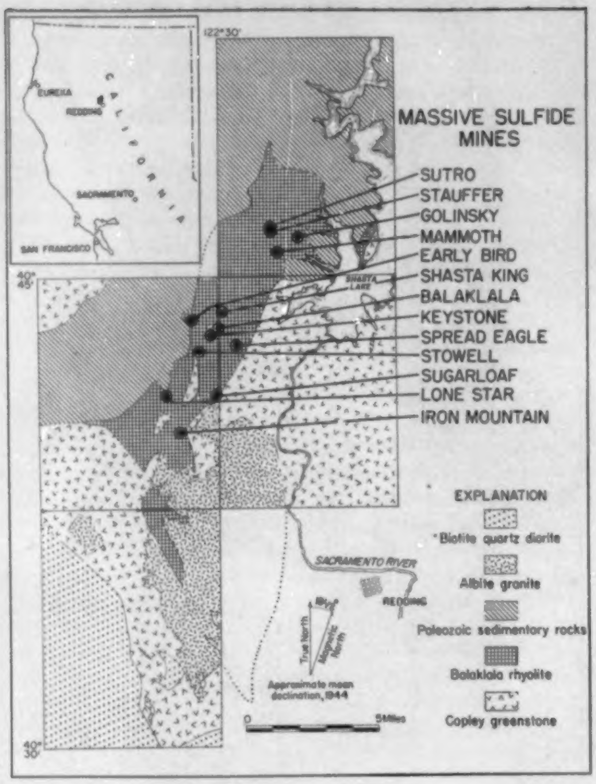

Fig. 2—Location and generalized geology of the West Shasta copper-zinc district, California.

of coarse to fine-grained rhyolite pyroclastics, which together form a broad elongate volcanic pile, shown diagrammatically in Fig. 4. Although the rocks of the Balaklala include many lithologic types that can be mapped separately, they are almost identical chemically and petrologically. The mode of formation of the rocks and their reaction to the secondary processes to which they have been subjected account for the difference in appearance between one rhyolite and another. Where the rhyolites are fresh and are not deformed they are typically hard light gray or light green felsitic rocks, many containing megascopically visible quartz and feldspar phenocrysts. In different flows the number of phenocrysts and their sizes differ, and this feature, in conjunction with bedded pyroclastic rocks, has been used to distinguish flows that are otherwise identical.

The Balaklala rhyolite is a complexly interlayered broad volcanic pile that is probably 3500 ft thick in the central area of rhyolitic eruption. It is largely submarine in origin, although part may have extended above sea level locally as volcanic islands. The volcanic rocks were extruded over a period of time from numerous widely separated vents, yet a recognizable stratigraphy is present in the pile, and the volcanic rocks that were extruded early in the sequence can be mapped separately from those extruded later. Geologic mapping has shown that there was progressive crystallization of phenocrysts of quartz and feldspar in the magma chamber from which the flows were derived, and although there are many exceptions and reversals, the earliest and most widespread flows were nonporphyritic, the next contained phenocrysts of quartz 1 to 4 mm in diam, and the latest contained coarse quartz phenocrysts over 4 mm. The Balaklala is subdivided into three units; the nonporphyritic rhyolite is the lowest, the rhyolite with medium phenocryst the middle, and the rhyolite with coarse phenocryst the upper unit. These lithologic types have been used as map units, although it was found that they must be used with caution. For example, later rhyolites intruded earlier flows as dikes and sills and are not always distinguishable; some vents were quiescent for long periods, with the result that some types of flows are absent in certain areas; many flows are long but limited in breadth; and lavas from different vents interfinger.

A determination of the stratigraphic sequence in the rhyolitic volcanic pile was essential to the study of the ore controls, as orebodies were found to occur within a limited stratigraphic range. The stratigraphic horizon, Fig. 4, in which ore was later deposited is in the upper part of the middle unit of the rhyolite.

The Kennett formation of Middle Devonian age was deposited around and possibly over the rhyolitic volcanic pile, Fig. 5. The Kennett is composed predominantly of black siliceous shale and limestone, but gray shale and rhyolitic tuff are present. The rhyolitic tuff is interbedded with shale, and locally it grades downward through a transition zone to rhyolitic tuffs that at places form part of the upper unit of the Balaklala rhyolite.

The Bragdon formation of Mississippian age apparently overlies the Kennett formation conformably in the mapped area, although outside this area warping seems to have uplifted part of the Kennett, and an erosional unconformity separates the Bragdon from the Kennett. The Bragdon formation is largely composed of shale, but it also contains beds of conglomerate, grit, and sandstone.

The Mule Mountain stock of albite granite intrudes the Copley greenstone and the Balaklala rhyolite in the mapped area, Fig. 2. It is a soda-rich siliceous intrusive rock composed principally of albite and quartz with minor amounts of epidote and, locally, hornblende. It is probably Late Jurassic in age but was intruded before the Nevadan orogeny was completed and is syntectonic.

The Shasta Bally batholith is a biotite-quartz diorite resembling parts of the quartz diorite of the Sierra Nevada. In the mapped area it intrudes the Copley greenstone, the Bragdon formation, and the albite granite, and to the west of the mapped area it

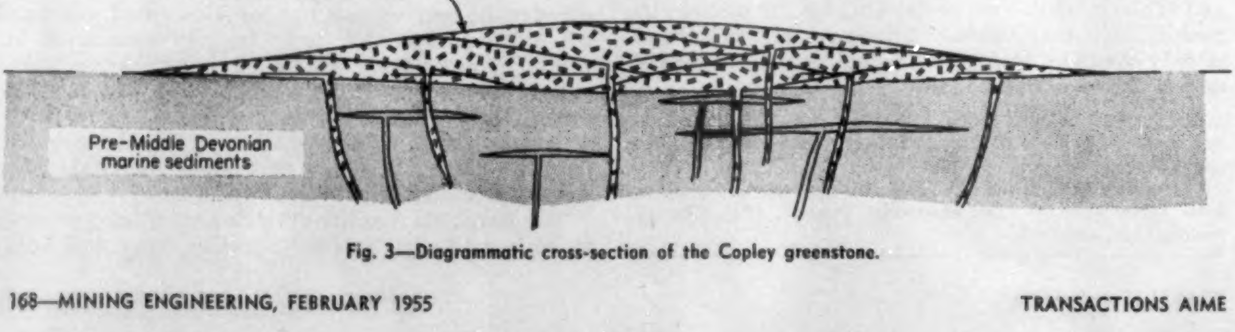

Fig. 3—Diagrammatic cross-section of the Copley greenstone.

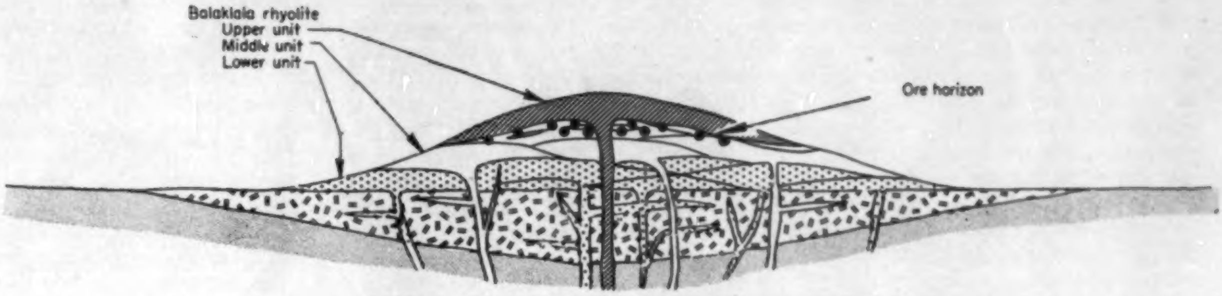

Fig. 4—Diagrammatic cross-section of the Balaklala rhyolite after deposition. Vertical scale exaggerated.

is overlain unconformably by rocks of Early Cretaceous age. A post-tectonic intrusive that was not deformed by the Nevadan orogeny, it is also of Late Jurassic age.

Fig. 6 illustrates diagrammatically how the present erosion cycle has cut through the thick cover of overlying shale and exposed the folded ore-bearing zone in deep canyons cut in the Balaklala rhyolite. Practically all the orebodies that have been found are those cropping out on the canyon walls and forming prominent gossans. A few concealed orebodies have been located by drilling, but much of the ore horizon, which in parts of the area is continuous between canyons under a cover of the younger rhyolite, has not been explored. This section shows in more detail the stratigraphic position of the ore zone in the upper part of the middle unit of the Balaklala rhyolite.

The Paleozoic rocks in the West Shasta copper-zinc district have been folded into a broad arch that contains many small folds and forms a broad, low anticlinorium; the axis trends N 15° E in the central part of the mineral belt, Fig. 7. The arch has a low culmination in the central part of the mining district; in the northern part of the district north of the Mammoth mine it plunges gently northward. The south end of the arch is disrupted by intrusives. Most bedding and flow contacts in the rocks of the mineral belt have gentle dips, generally less than 30°, and although the rocks are warped into many broad folds and domical structures, they are not strongly folded except in local areas. Folding is strongly developed where there is a great difference in the competence of adjacent rocks, as between conglomerate and shale, or where thin-bedded tuff of the Copley is interlayered with massive flows, if these beds are in zones of deformation. Variation in the amount and degree of folding is apparently due largely to lenticularity of flows and small intrusives and the consequent abrupt differences in competence. Regional stresses were transformed by a body of heterogeneous rock into many local stresses of different intensity and direction.

Many of the rocks in the mineral belt are weakly foliated or sheeted; strongly foliated rocks occur in local bands. The rocks range from schist and gneiss through moderately folded rocks with fracture cleavage to rocks that are practically undeformed. In parts of the district foliation is parallel to bedding; in other parts it cuts across the bedding. Where foliation is most intense primary textures and minerals are destroyed, and in these areas it is rarely possible to determine the relationship between foliation and bedding. The distribution of strongly and weakly foliated rocks depends in detail on the competence of the different rocks, in part on buttressing effects that determined the local stresses, and in part on the location of large masses of intrusive rocks.

Two types of foliation are present. One is a steeply dipping planar structure ranging from widely spaced jointing or closely spaced sheeting to fracture cleavage with the development of films of aligned secondary minerals along closely spaced planes. In this type of planar structure the rock between cleavage planes is unaffected. Flow cleavage, in which all the rock minerals have been oriented by stress, is developed locally along zones of intense movement, but such zones are rare in the minerals belt.

The second type of foliation common in the mineral belt is bedding-plane foliation developed by flexural-slip folding. Even though most of the folding is gentle, bedding-plane foliation is locally well developed, probably because of the great differences in competence in the layered rocks. This type of

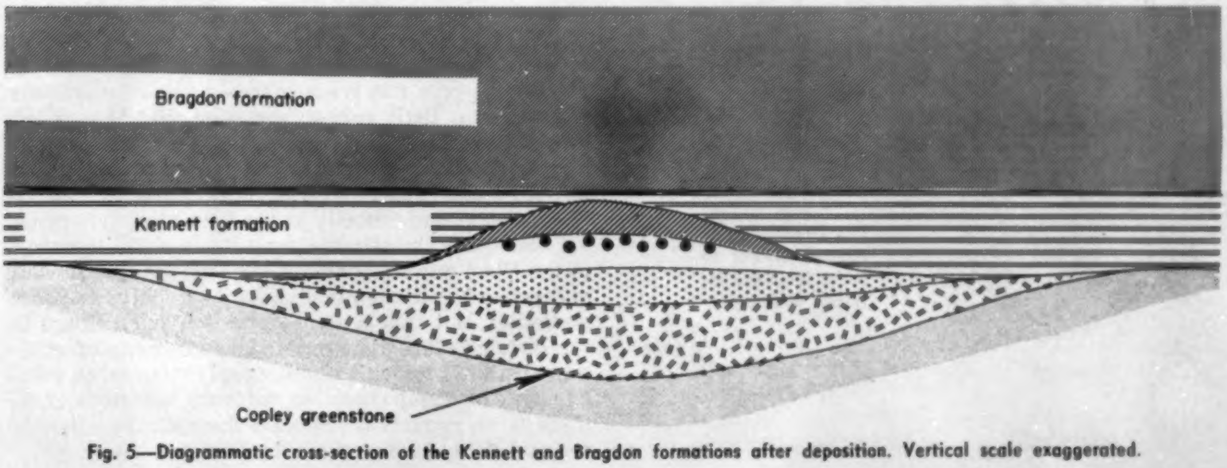

Fig. 5—Diagrammatic cross-section of the Kennett and Bragdon formations after deposition. Vertical scale exaggerated.

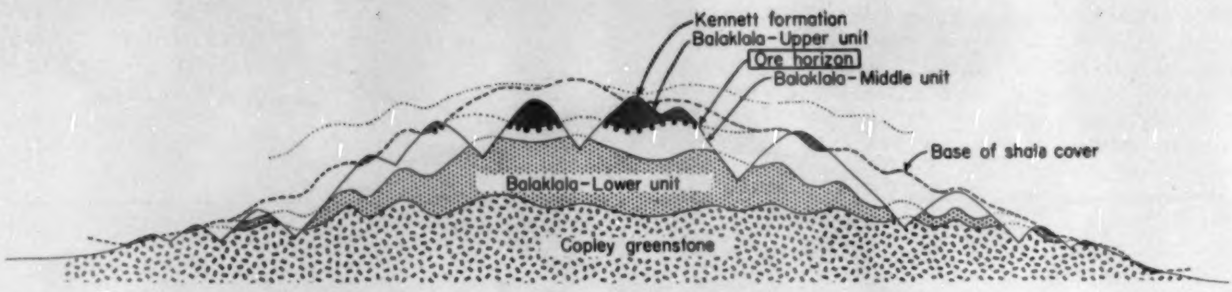

Fig. 6—Diagrammatic cross-section of the West Shasta copper-zinc district.

foliation develops in horizons that contain tuff and volcanic breccia, as these rocks are less competent than the flows. Adjustments between flows were taken up along the flow contacts rather than within the flows.

Planar structures are not evenly distributed in the rocks. A thick flow, such as the coarse-phenocryst rhyolite, acted as a competent body and was foliated much less than the group of lenticular flows and pyroclastic rocks underlying it. In addition, steep foliation is more strongly developed in some areas on the axes than on the flanks of folds. Thus under some conditions a body of foliated rock has a linear element, that is, only certain flows are foliated, and these are most strongly foliated along fold axes, forming an elongate body of foliated rock limited above and below by less foliated flows and plunging with the fold axes.

Production

Of the copper produced in California before 1946, 54 pct came from the Shasta copper-zinc district, the major part from the West Shasta area. Zinc production was small because most of the ore was direct-smelted and zinc was not recovered; only bodies of high-grade zinc ore were mined and treated separately as zinc ore. Gold and silver were recovered from gossan that overlies massive sulphide ore at the Iron Mountain mine and from sulphide ore that was smelted for copper. At the Iron Mountain mine 3,600,000 tons of massive pyrite have been mined for its sulphur content alone, but sulphur was not recovered at any of the other mines. Several thousand pounds of cadmium was recovered from zinc-rich ore at the Mammoth mine.

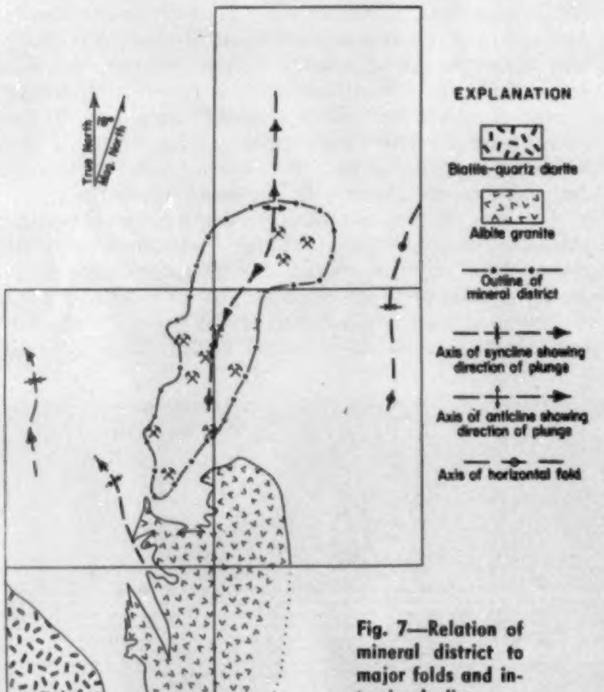

Fig. 7—Relation of mineral district to major folds and intrusive bodies.

Character and Distribution of Ore Deposits: The copper-zinc orebodies of the West Shasta district are bodies of massive pyrite that contain chalcopyrite and sphalerite and minor amounts of gold and silver. The most striking features of the ore are its uniformity, its lack of megascopic gangue minerals, and the sharpness of its boundary with barren or weakly pyritized wall rocks. The ore has a brassy appearance, and some large bodies contain as little as 3.5 pct of insoluble material. In most orebodies the massive sulphide is separated from barren wall rock by a thin selvage of gouge; gradational contacts between ore and wall rock are very rare. Most orebodies of the district are lenticular flat-lying, their greatest dimensions in a horizontal plane. Several are saucer-shaped, one is domal, and one synclinal. Steeply dipping orebodies are not characteristic of the district, but they do occur in the Hornet orebody at the Iron Mountain mine, at the Golinsky mine, and at the Sutro mine.

All the bodies of massive sulphide contain some copper and zinc, but massive pyrite that contains only small quantities of copper and zinc occurs at many of the mines. At present the only value of such low-grade pyritic bodies is their sulphur content, as the iron-rich residue from roasting has not been utilized. Massive pyrite with little copper and zinc occurs both as discrete orebodies and as low-grade parts of orebodies containing minable amounts of copper and zinc. In some mines the upper or lower part of a flat-lying massive sulphide orebody contained so little copper and zinc that it was left in place. Massive sulphide ore with a high percentage of zinc occurs mainly at the Mammoth mine, although zinc was recovered by flotation from parts of the Richmond orebody at the Iron Mountain mine.

Some disseminated chalcopyrite in rhyolite occurs in the No. 8 mine orebody of the Iron Mountain mine and in the Balaklala mine below the massive sulphide orebodies, but this type of deposit is exceptional in the district. The disseminated ore consists of chalcopyrite and pyrite, in about equal amounts, as veinlets and disseminations in siliceous schistose rock; there is no gradation between the massive sulphide ore and the siliceous disseminated copper ore.

Pyritization is widespread in the district. Bodies of weakly pyritized rhyolite extend for several square miles, and practically all the rhyolite along the mineral belt contains some pyrite. Massive sulphide orebodies that contain copper and zinc occur as sharply bounded discrete bodies in the broad zone of pyritization. There is no correlation, in many instances, between the amount of scattered pyrite in the zone of pyritization along the mineral belt and the location of minable bodies of massive sulphide ore, and there is normally no gradation between pyritized rock and massive sulphide. Gold, silver, copper, and zinc values are almost entirely confined to massive sulphide bodies as far as is known.

Figs. 8, 9, and 10 illustrate the typical shapes of orebodies in the West Shasta district. The Iron Mountain mine shown in Fig. 8 contains the largest orebody in the district. Before faulting and erosion, this was a single body of copper and zinc-bearing massive pyrite about 4500 ft long, probably containing 25,000,000 tons of sulphide ore. The orebody is a sharply bounded sulphide mass that appears to have formed by the replacement of sheared porphyritic rhyolite. The wall rock is only locally pyritized. The orebody is in part synclinal, as at section A-A', and follows the folded structure of the enclosing rhyolite flows and pyroclastics. It is cut into separate blocks by post-mineral transverse faults, as shown in the longitudinal section C-C'. The orebody is in a pre-mineral fault zone, probably a feeder channel, along which some post-mineral movement has occurred.

The Shasta King mine shown in Fig. 9 is an example of a saucer-shaped orebody. The massive sulphide orebody lies immediately beneath a layer of bedded tuff and volcanic breccia and apparently replaces a thin flow of porphyritic rhyolite. It is conformable to the bedding in the lavas and pyroclastics. The saucerlike form is shown in the two intersecting cross-sections. No feeder channel was recognized at the Shasta King mine.

The ore at the Mammoth mine is on a dome or arch, Fig. 10. The structure contour map in the upper part of the illustration is drawn on the contact between the upper and middle units of the rhyolite and shows the correlation between local domal areas at this contact and the location of much of the ore. The northeast trending fault zone shown on the map has both pre-mineral and post-mineral movement, and at depth it may have been the feeder channel, although at the stratigraphic level of the orebodies there is apparently no close relation between the fault zone and the orebodies. The distribution of small orebodies along the northwest-trending fault suggests that this fault also may be pre-mineral in age. The cross-section A-B-C-D illustrates the stratigraphic control of ore deposition. At this mine ore was deposited just below the contact between the upper and middle units of the Balaklala rhyolite.

Relation of Deposits to Stratigraphy: Minable orebodies found thus far in the West Shasta district are at the same stratigraphic horizon in the upper part of the middle unit of the Balaklala rhyolite throughout the district, Fig. 11. Ore is known to occur through a stratigraphic thickness of 600 ft in the Balaklala rhyolite at the Iron Mountain mine, and it is possible that locally the favorable zone may have a greater thickness. The top of the favorable zone is the base of the upper unit of the Balaklala rhyolite, which is composed of coarse-phenocryst rhyolite or tuff, but the lower limit is not marked by distinctive flows. The upper part of the middle unit is a group of discontinuous flows and lenticular beds of coarse and fine pyroclastic rocks. The heterogeneous nature of this material is such that the detailed stratigraphy at each mine is unique, yet the fact that this heterogeneous group is capped by a recognizable unit over much of the district makes it possible to locate the ore zone at one general horizon with a fair degree of certainty.

Exploration done thus far has shown that pyritization is fairly continuous along the mineral belt in this limited stratigraphic zone, even though bodies of massive sulphide ore are scattered. Many exploratory holes have disclosed heavily pyritized rock in the favorable stratigraphic zone at considerable distances from known ore, and orebodies have been located by systematical drilling of areas that contained favorable zone.

At some places the character of the rock that was replaced to form orebodies can be determined; the favored host rock for orebodies appears to be porphyritic rhyolite that has quartz phenocrysts of 2 to 3 mm, particularly where this rock is overlain by thinly bedded tuff or fine pyroclastic rocks.

Relation of Deposits to Folds and Foliation: Both individual orebodies or groups are concentrated on or near the axes of broad folds. Orebodies formed in anticlines and in synclines, although synclinal bodies predominate if basin-shaped structures are

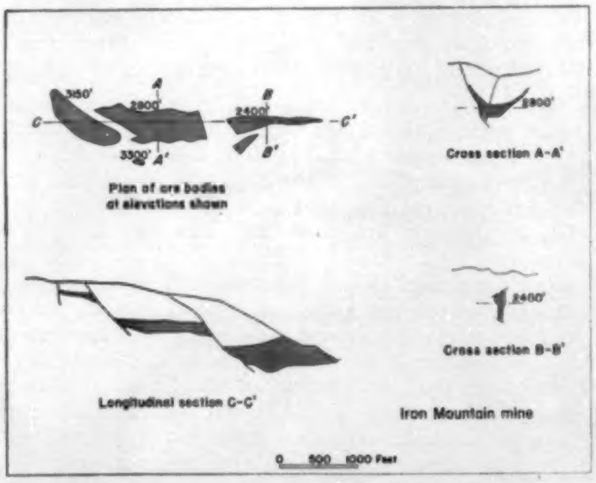

Fig. 8—Plan and sections of the Iron Mountain mine.

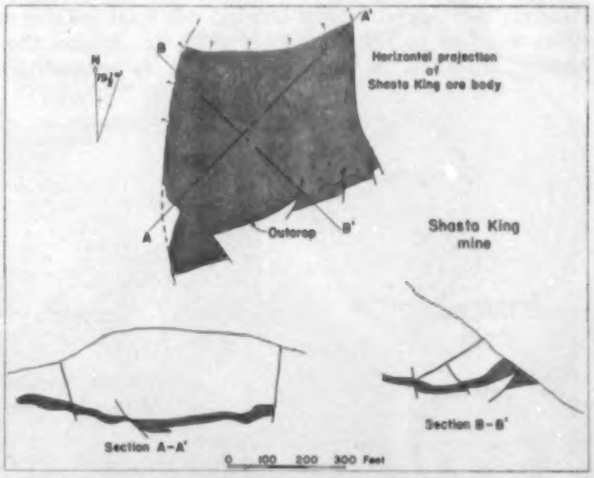

Fig. 9—Plan and sections of the Shasta King Mine.

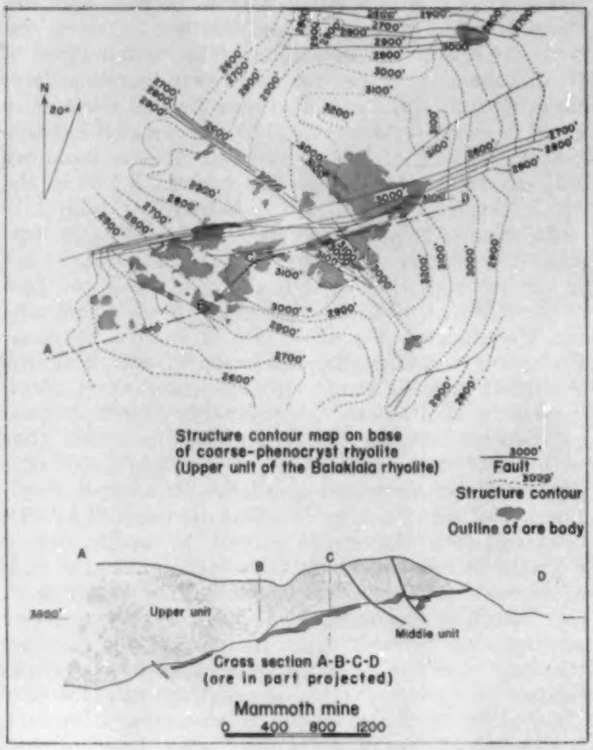

Fig. 10—Plan and section of the Mammoth mine.

included with the synclines. The Richmond orebody of the Iron Mountain mine, the orebodies of the Balaklala, the Shasta King mines, and probably the orebody of the Early Bird mine are examples of those formed in synclines or basin-shaped warps. The broad ore zone at the Mammoth mine is on an arch or dome-shaped structure, and at many localities mineralization favored the crests of small folds. The orebodies at the Sutro and Keystone mines and the Old Mine orebody at the Iron Mountain mine appear to be on the flanks of folds.

Probably because the folds are broad, foliation is not strongly developed except locally in the mineral belt, but the two types of foliation developed in flexural-slip folding, bedding plane foliation and fracture cleavage, have had considerable effect on ground preparation prior to ore deposition.

Bedding plane foliation is concentrated along layers of bedded pyroclastic material and is most strongly developed where bedded material between flows is a few inches to a few feet thick. Where the bedded material is thicker, foliation is commonly limited to a zone at the top or bottom of the bed or is distributed as interbed movement along planes throughout the bed.

Poor to well-developed steep fracture cleavage in competent layers occurs in conjunction with bedding-plane foliation, ranging from little more than subparallel jointing to foliated rocks with reorientation of minerals along cleavage planes. The steep fracture cleavage is very rare in the coarse-phenocryst rhyolite of the upper unit, but locally it is well developed in the flows underlying this rock, particularly in flows capped by a fairly continuous horizon of pyroclastic material. The coarse-phenocryst rhyolite is the most competent rock in the Balaklala, but the nonporphyritic rhyolite of the lower unit and the medium-phenocryst rhyolite of the middle unit are also competent rocks except where they contain layers of bedded pyroclastic material. In the folding along the mineral belt steep fracture cleavage developed in the lower and middle units, particularly along axes of folds, but such cleavage is rare in the upper unit. Bedding-plane movement with accompanying bedding-plane foliation was concentrated in the layers of bedded pyroclastics in the upper part of the middle unit, see Fig. 12. The intersection of steep fracture cleavage with flat foliation controlled by bedding formed a zone of fractured rock along the crest or trough of a fold, under a relatively impervious cover of unfractured rock of the upper unit.

Features Controlling Ore Deposition: Three main base-metal ore controls can be recognized in the copper-zinc district: 1—the stratigraphic control within the Balaklala rhyolite, 2—the structural control by folds and foliation, and 3—the feeder fissures along which the solutions ascended. All the base-metal orebodies in the West Shasta district are in the Balaklala rhyolite and are further restricted to the upper part of the middle unit of the rhyolite. Orebodies are localized along broad folds and warps, although they show little preference between anticlines and synclines. The mineral belt as a whole follows the trend of a series of broad folds that constitute an anticlinorium, and there appears to be a correlation between the culmination of the anticlinorium and the central part of the mineral belt. In detail, individual bodies appear to be localized along minor folds or warps. Both bedding-plane foliation and fracture cleavage are related to these folds. The intersection of steep fracture cleavage with gently dipping bedding-plane foliation provided a shattered area with a relatively impervious capping that localized some orebodies. Steep fracture cleavage may have acted as a collecting agency

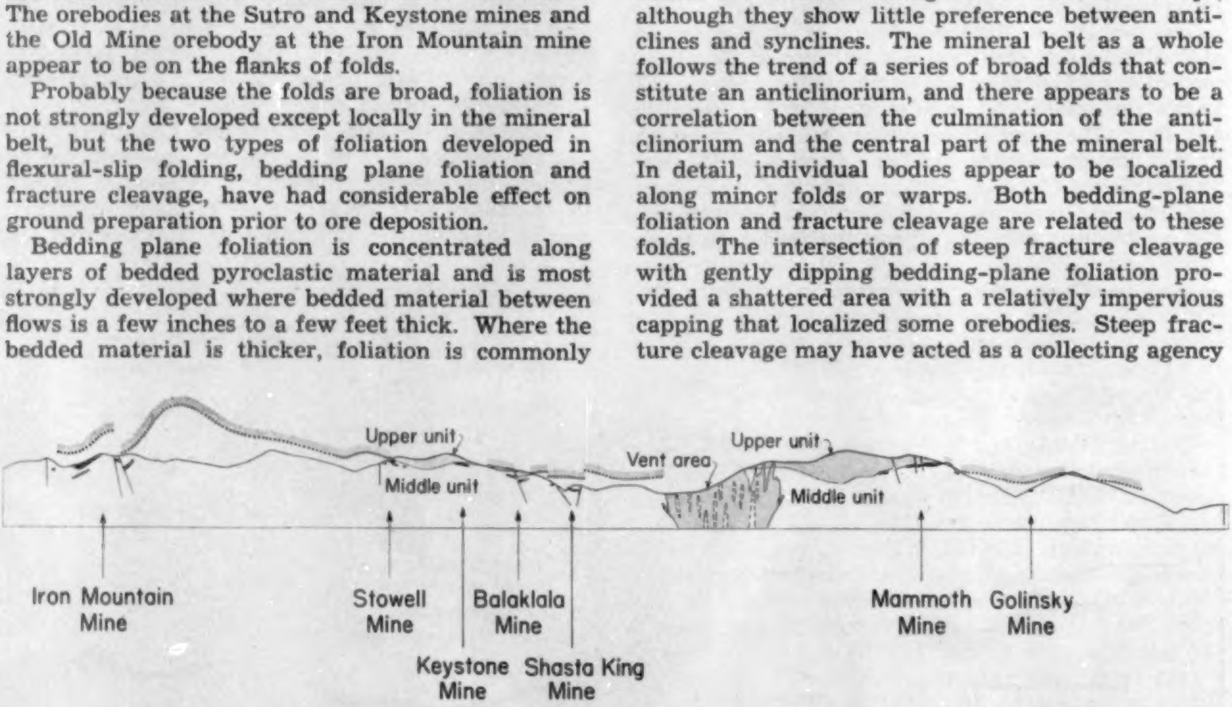

Fig. 11—Longitudinal section along the central part of the mineral belt.

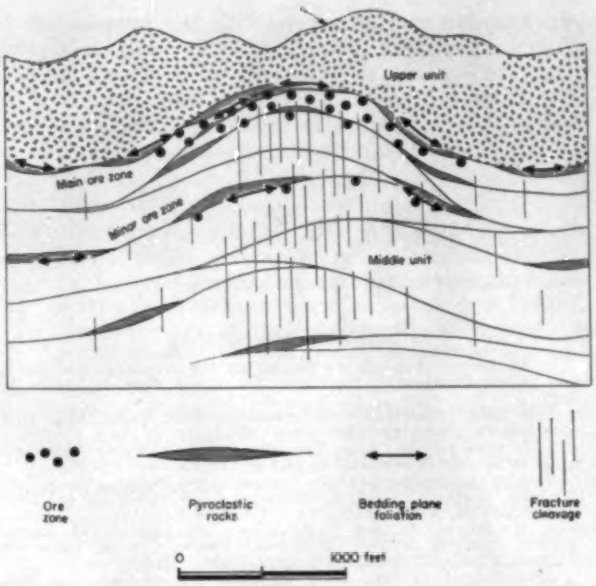

Fig. 12—Diagrammatic drawing showing relation between the ore zone and controlling geologic structures.

for dispersed rising solutions that were then channeled laterally along the gently plunging intersection of flat bedding-plane foliation and steep fracture cleavage.

In spite of the close correlation between the stratigraphy and the location of ore shown in Figs. 6, 11, and 12, the actual control of ore deposition is apparently due to structural features rather than to the presence of flows at one horizon that were by nature particularly susceptible to replacement. Correlation between stratigraphy and ore is due to the fact that during folding, stratigraphic features, mainly differences in competence and reaction to folding, controlled the location of fractures that formed solution channelways, and these in turn localized the orebodies.

Some faults are pre-mineral in age and have acted as channelways for ore-bearing solutions. They generally cut the folds and the foliation at a considerable angle and were influential in localizing orebodies in certain parts of the folds.

A conjunction of the three types of ore controls was probably a pre-requisite for the formation of a major orebody, but they occur in conjunction because they are interrelated. Lenses of pyroclastic material initiated local folding because in contrast to massive flows they form less competent layers and because a change in dip, along the boundaries of a lens in this instance, is a favored locus for the start of folding. Movements along the bedding plane were most pronounced in folded areas, and fracture cleavage developed mainly in the axial regions of folds. The relationship, if any, between the location of feeder fissures and the other ore controls is not known.

Exploration Possibilities: Many favorable areas where the ore horizon has not been eroded remain to be explored in the West Shasta district, but other large areas can be eliminated from exploration for geologic reasons. Areas worthy of exploration are those that contain the middle unit of the Balaklala rhyolite within the main northeast-trending mineral belt; those that can be eliminated are areas in which the middle unit has been removed during the present erosion cycle or is missing because of original lenticularity and nondeposition.

The stratigrahic sequence is the principal feature used in delimiting areas in which new orebodies may be found. All known ore in the district is in the middle unit of the Balaklala, generally in the uppermost part of the middle unit, particularly where this unit contains much bedded pyroclastic material. No minable bodies have been found in the lower unit, although it is heavily pyritized at some places. No orebodies and practically no mineralization are present in the upper unit of the Balaklala, but areas in which the upper unit is present are favorable because the middle unit can be assumed to be present below. No massive sulphide deposits have been found in the underlying Copley greenstone. However, the Copley is exposed in very few places along the mineral belt, and the possibility of copper deposits on the chloritic rocks of the Copley along feeder channels should not be ignored. The Copley is deeply buried along most of the mineral belt, and the location of feeder channels in certain areas is suggested but not proved.

No horizontal controls for ore can be used with certainty to eliminate areas where the middle unit is known to be present, for although most ore occurs along the crests or troughs of gentle folds or warps, some occurs on the flanks of folds. The latter localities, though probably less favorable, cannot be eliminated from exploration.

The most favorable areas, shown in Fig. 13, are: 1—areas between known orebodies along the trend of the mineral belt, particularly where hydrothermal alteration and pyritization is present along the crests or troughs of folds; 2—extensions of folded structures beyond known orebodies; and 3—areas along the trend of fissures that appear to be main feeder channels, where these fissures cross folded structures in the productive horizon.

Permissive areas, although less favorable than those described above, are where the middle unit

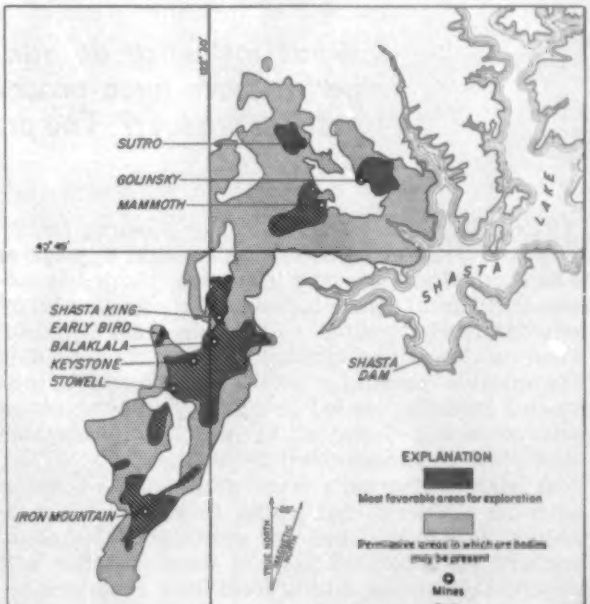

Fig. 13—Recommended areas for exploration in the West Shasta copper-zinc district.

of the Balaklala rhyolite is thin or where some of the upper part of the middle unit has been eroded. Areas where the rocks are closely folded are not as favorable as areas in which dips are gentle.

On the basis of the ore controls that have been described it is now possible to delimit areas in which ore may be found and to eliminate others from consideration. Fig. 13 covers such a large area that it does little more than indicate where geologically guided exploration might find ore, but it does indicate the great extent of the favorable areas that have been only partly explored. Although some prospect drilling has been done near known orebodies, many areas with no surface indication of ore deserve exploration. In this district there may be little or no surface indication of large bodies covered by only 100 to 200 ft of rhyolite. Thus it is imperative in the search for concealed ore to recognize the stratigraphic position of the different types of rhyolite. It must be determined in which areas the favorable ore horizon has been eroded, in which areas it is present under a cover of younger rocks, and at what approximate depth. When the favorable horizon has been located and favorable structures determined, it must also be recognized that the solution channels formed by the intersection of gently dipping bedding-plane foliation and steep fracture cleavage in most cases have a gentle plunge. Solutions probably traveled considerable distances horizontally along these channelways, and areas of strong hydrothermal alteration may mark only the outlet of a solution channel.

With these cautions in mind, it is predicted that new orebodies are yet to be found in the West Shasta district if exploration is based on the ore controls that have been described.

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Processing Perlite— The Technologic Problems

by Robert H. Weber

What influence do variations in commercial-grade perlite have upon processes used to prepare a marketable product? The problems are summarized here.

INCREASING acceptance of perlite products, chiefly in the fields of lightweight structural aggregates and thermal and acoustic insulation, has led to expanding market demands that have encouraged many new producers to enter the field. In some instances, however, the failure of these producers to anticipate the variable response of perlite to conventional processing methods has led to difficulty in establishing an economical flowsheet by which a predictable specification product could be obtained.

It is not the writer's intent to provide a solution to these problems, but rather to summarize their nature. It is to be hoped that members of the industry who have hurdled some of these obstacles will document solutions arising from their experience.

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Discussion on this paper, TP 3971H, may be sent (2 copies) to AIME before April 30, 1955. Manuscript, May 17, 1954. El Paso Meeting, October 1953.

In this treatment the term *perlite* will not be restricted to the petrographic definition but will apply to all volcanic glasses. Expansible obsidians and pitchstones are accordingly included in this broader industrial classification.

Chemical Properties: Although perlite has been reported to range in composition from that of rhyolite to that of andesite, it is probable that most of the glasses have the composition of rhyolite. When recalculated to an anhydrous basis, the five analyzed glasses from New Mexico, representing five distinct physical types from widely separated deposits, show an amazingly uniform oxide composition. Water content is the only major compositional variable; total water ranges from a low of 0.37 pct (nonexpansible obsidian) to a high of 8.95 pct (expansible pitchstone). The wide range in expansion characteristics exhibited by these samples cannot be related to significant variations in the composition of the nonvolatile fraction and has been only partially correlated with variations in water content.

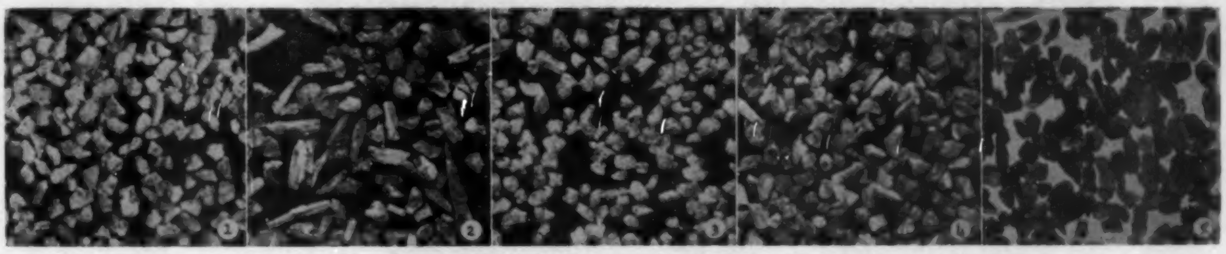


Fig. 1—The —14 + 28 fractions of crushed perlite aggregates. Variations in particle shape and texture reflect inherent differences in the physical characteristics of the crusher feed. Perlite textural types illustrated are: 1—pumiceous, non-perlitic; 2—*needle* type, prominently perlitic; 3—fine-grained, highly perlitic; 4—*onion skin* type, coarsely perlitic; and 5—pitchstone, very slightly perlitic. X4. Area reduced approximately one-half.

Physical Properties: The physical properties of perlite exhibit a wide range of variation, from massive and vitric to cellular, granular, and fragmental; many hues of color from grayish white to black; and lusters that are vitreous, pearly, pitchy, or resinous. Although they may be grouped into several distinct textural types, these types are not distinguished by sharp boundary differences but are completely intergradational. Several textural types may be closely associated in a single deposit.

Some generalized relationships are apparent between milling and expansion characteristics and the physical properties of the glass. Certain combinations of these properties, particularly texture, luster, and color, together with the water content, may aid in an approximate evaluation of the commercial potentialities of the glass. Thus fragmentation characteristics in milling and a differentiation between *lively* and *dead* glasses may be anticipated to an approximate degree. Unfortunately, the accuracy of such predictions is based upon personal experience, and some glasses will prove unpredictable.

Mining: The low unit value of perlite crude limits the scope of practicable mining methods. Deposits not amenable to open-pit exploitation have been avoided by most operators. Cost limitations have also eliminated most deposits in which variations in quality necessitate highly selective mining.

Some deposits are sufficiently fractured or friable to permit mining by use of a ripper with tractor and carryall or by bulldozer alone. Where drillhole blasting is required, care must be exercised in selecting an explosive and a blasting method that does not overbreak the ore, producing an excess of fines. Despite its hardness, most perlite drills easily and rapidly and breaks well under blast owing to its brittleness. Secondary blasting is rarely necessary.

Milling: Inherent differences in the physical character of perlites from different deposits have indicated the inadvisability of selecting a crushing and sizing circuit merely because it has proved successful at another deposit. The design of a milling flowsheet requires thorough preliminary testing, beginning with laboratory evaluation and culminating in tests on a pilot mill scale.

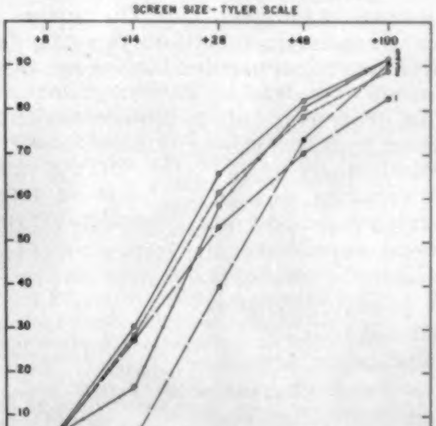
Several fundamental problems that must be resolved in this stage of processing may be stated as follows:

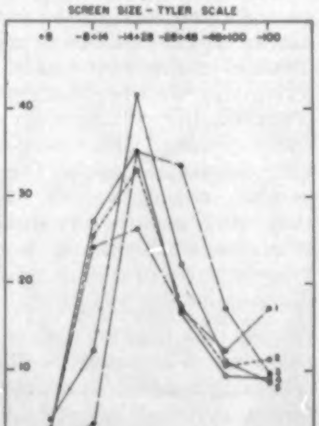
1—crushing to produce particles approximating a cubic shape, 2—crushing to produce the required particle size gradation, and 3—sizing to specification particle gradation.

Current crushing practice involves one-stage treatment in some plants, whereas others utilize multi-stage flowsheets. Impactors, operating at a high percentage recycle, have received widespread acceptance among users of one-stage systems. Multi-stage operations have favored jaw crushers in the primary stage. Various machines have been successfully utilized in the secondary and subsequent stages, including gyratory and cone crushers, rolls, impactors, and in at least one operation, a rod mill. Removal of the undersize by scalping screens between each stage aids in minimizing the proportion of wasted undersize produced. Dry processing is vastly preferred to wet treatment owing to the high costs incurred in drying the final product, and the difficulty of wet sizing in the finer ranges.

Although it is desirable in crushing to produce particles that approach a cubic shape, this is difficult to accomplish in commercial practice. Most perlites show tendencies to break in preferential directions along pre-existing fractures or planar structural elements. Some variations resulting from

Figs. 2 and 3—Screen analyses of five types of crushed crude perlite aggregate. Samples were prepared by passing through a laboratory jaw crusher followed by further reduction in laboratory rolls, without intermediate sizing. Dust losses were ignored. Represented perlite textural types are: 1—pumiceous, non-perlitic; 2—*needle* type, prominently perlitic; 3—fine-grained, highly perlitic; 4—*onion skin* type, coarsely perlitic; and 5—pitchstone, very slightly perlitic. It will be noted that the graphs of individual samples are divisible into two groups, each of which has a characteristic size distribution pattern. Samples 1, 2, and 4 form one group, and samples 3 and 5 form a second group.





differences in physical character are illustrated by Fig. 1. Highly perlitic types (*onion skin*) tend to break into curved spalls and rounded core kernels in the coarser fractions. Thin flakes may predominate in the finer fractions. *Needle-type* perlites, which are simply variants of the onion skin type in which the perlitic fractures have a pronounced parallel orientation in one direction, produce elongate splinters that are very difficult to size accurately. The massive, texturally non-perlitic glasses have a pronounced tendency toward conchoidal fracture that favors the production of concavo-convex chips or shards. It is therefore evident that no single scheme of crushing is equally adaptable to each of the various textural types of perlite.

Thorough sizing is essential to the production of a predeterminable expanded product owing to the multiplication of particle size resulting from expansion. Most glasses tend to yield a crushed product high in the fine fraction, a large portion of which is —100-mesh and accordingly wasted by removal from the sized product. The variation in proportion of size fractions yielded by several types of perlite is shown in Figs. 2 and 3, which represent the results of a single crushing test of each type.

Multiple deck vibrating screens are largely preferred for the sizing operations, although at least one operator has found a spiral rotary machine advantageous in securing maximum passage of undersize particles. Sizing in the finer fractions is commonly accomplished with air separators.

Dust nuisances have been reduced by the use of covered screens and the maintenance of dust producing points under vacuum, the dust fraction being collected by cyclones.

Removal of excess free moisture is usually desirable. In addition to facilitating sizing and handling operations, the dried product will permit freight cost savings in shipment to the consumer and may give better performance in the expansion process. Both rotary kilns and stationary flash driers have been used for this purpose.

Expanding: The expansion process and characteristics of furnace design have been more fully treated in the technical literature than have the other phases of perlite processing. The reader is accordingly referred to the thorough discussion by Murdock and Stein[1] for an analysis of furnace design features, and to King et al.[2] for thermal expansion and energy requirements data. Calculations of energy requirements should, of course, include a correction for the low thermal efficiency of conventional furnaces.

To date there has been no scheme devised for accurately predicting the furnace behavior of a given perlite on the basis of its chemical composition and physical character. Crude differentiation between *lively* and *dead* perlites may be made by an evaluation of the megascopic physical character and water content. Laboratory expansion tests will provide details relative to the practical ranges of expansion, ranges of bulk density, resultant particle shape and texture, compressive strength, proportion of nonexpansible waste, and preheat requirements. The difficulty of duplicating laboratory results with the commercial type of furnace, however, dictates the need for final testing on a pilot mill scale.

Perlite is sensitive to slight changes in furnace operating conditions; hence the use of only a manual control system is largely precluded. Interlocked automatic systems that provide close instrumental control of fuel-air ratio, upheat rate, temperature level, kiln pressure, and feed rate are now considered a necessity rather than a luxury.

Difficulty has been encountered in obtaining uniform expansion of both fine and coarse particles in the graded feed. Treatment favoring optimum expansion of the fine feed does not fully expand the coarser fraction, whereas treatment geared to optimum expansion of the coarser fraction may result in overheating the fine fraction, thus promoting the formation of kiln rings and the collapse of expansion cells in the product. Various techniques have been devised to resolve this problem; one operator uses two rotary kilns in series, whereby the fine particles expanded in the primary kiln are removed from the circuit by cyclones and the coarser particles are finished in the secondary kiln.

Miscellaneous Problems: Perlite, in both crude and expanded form, is highly abrasive. Careful consideration should be given to this property in planning equipment requirements, operational techniques, and repair and replacement costs.

Disposal of the excess —100-mesh fraction of both crude and expanded products has proved troublesome at many plants. There is currently little market outlet for this material; hence it is largely wasted to the dump, where it constitutes an objectionable dust hazard in urban areas and may occupy ground that could be put to more beneficial use. If the costs of this waste which accrue from mining, transportation, crushing, sizing, drying, and expansion (in the case of the finished product) are considered, and if it is noted that the waste usually constitutes from 5 to 10 pct or more of the milled product, and a significant fraction of the expanded product, it is evident that efforts to develop markets for its disposal would be well expended. Possible uses include ceramic glazes,[3] glasses (here is a field for applied research), pozzolanic concrete additives, abrasives, filter aids, and fillers.

Conclusions: Great advances have been made in the technology of perlite processing during the past several years, but relatively few of these advances have been adequately documented in the available literature in response to growing public interest.

The variable character of perlite and its somewhat capricious response to conventional process practice precludes a standardization of treatment which parallels that of many industrial mineral operations. Thorough testing from laboratory through pilot mill scale is considered a prerequisite to a full evaluation of equipment needs and treatment procedure.

The current trend is directed away from the production of a multipurpose aggregate and toward an increasing array of specification products tailored to meet industry standards and the individual requirements of consumers. This factor, coupled with inherent variations likely to be found in the character of the crude, clearly indicate that the plant should be designed for rigid control, yet permit a considerable range of flexibility of operational scope.

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aime NEWS

President Reinartz Visits Nevada Group

Leo F. Reinartz, AIME President, addressed 145 members and guests of the Nevada Section of the Institute at a dinner meeting held at the Flamingo Hotel in Las Vegas. John B. Zadra, Chairman of the Nevada Section, greeted the members and guests and announced that the Southern Nevada Subsection of the Nevada Section had been reactivated. Victor E. Kral, Secretary, Nevada Section, announced the names of the 1955 officers of the Nevada Section who are: F. A. McGonigle, Chairman; James R. Wilson, Vice Chairman; John N. Butler, Secretary-Treasurer; Hubert B. Chessher, Chairman Membership Committee; J. B. Zadra, Section Delegate; James R. Wilson, First Alternate; Vernon E. Scheid, Second Alternate; and the newly selected officers of the reactivated Southern Nevada Subsection who are: Sidney McCarroll, Chairman; James Orr, Vice Chairman; Ray Lundquist, Secretary. Southern Nevada now has attained a prominent position in the mineral industry and many engineers are employed in that area. The evening meetings of the Southern Nevada Subsection are tentatively planned to be held bimonthly and should be well attended.

Mr. Reinartz addressed the Henderson Rotary Club with an informative talk on *Steel and the Steel Industry*, on Friday afternoon. In discussing the sources of iron ore that the U. S. must rely upon in case of another war, it was noted that various companies will soon have spent about one half million dollars on the concentration of low grade iron ores such as taconite. This will be one of our major sources of iron ore in the future.

About 50 members made inspection tours to the various installations on Saturday. The trips were arranged so that they were of interest to all the engineers—mining, geological, metallurgical, and chemical. The largest group inspected the Henderson plants. They were first taken through Western Electro Chemical's plant. L. G. Immonen and other members of the staff showed and explained the various steps in the making of electrolytic manganese dioxide; from the leaching plant through the electrolytic cells and the final grinding and processing of the high grade product.

P. B. Peterman of Pioche Manganese then took the group through the plant now making ferromanganese where they witnessed a tapping of the melt. Lunch was served the group by the courtesy of the various Henderson plants and they then visited the operation of Manganese Inc. Sidney McCarroll and other members of the staff took the group around the entire operation; the pit where Isbell Construction Co. is doing the mining, the new flotation mill which is handling 1200 tons of ore per day, and the kiln which makes a high grade manganese oxide clinker.

Another group visited the operation of Molybdenum Corp. of America at Mountain Pass, Calif., 60 miles south of Las Vegas. This is the much-publicized deposit of baestnesite, containing large quantities of rare earth elements.

Plan to Publish AIME Directory

Following careful study and review of various proposals and suggestions for publishing a directory or directories of AIME members, the following plan has been approved:

Names, positions, and addresses of all AIME members as carried on the master file cards of the Institute on Feb. 1, 1955 will be used. Both an alphabetical and geographical listing will be typed and printed by offset. The geographical listing will, as in the last Directory, indicate which members are available for consulting work, and will also indicate the field of their primary and subsidiary professional interests. It is planned that the Directory may be available not later than June. A supplement to the Directory, similar to that published in the last four years, containing a list of the officers, Branches, Divisions, Local Sections, Student Chapters, committee personnel, and the Bylaws of the Institute will also be published, possibly as part of the Directory.

Following the procedure now adopted by the ASCE, ASME, AIEE, and AIChE, the Directory will be mailed only to those members in good standing who request it. A coupon will be published in the March issues of the Institute's journals, offering the Directory. This coupon, when filled out and returned, will serve as the address label for mailing the Directory. Members should request copies of the Directory by filling out this coupon.

EJC to Sponsor Nuclear Congress

Engineers Joint Council is sponsoring a Nuclear Congress to be held in Cleveland Dec. 12 to 16, 1955. All phases of nuclear science and engineering will be covered except restricted matter principally of a military nature. It is planned to make the meeting a worthy successor to, and expansion upon, the meeting so successfully conducted by the American Institute of Chemical Engineers at Ann Arbor last year. Most of the constituent societies of EJC will take part, as will many others, notably the newly formed American Nuclear Society. Well over 1000 are expected to attend.

W. V. Smith Joins Publications Staff

Beginning the first of the year, Warren V. Smith took over the job held by Frederick A. Stanley the last two years as Eastern Advertising Manager for MINING ENGINEERING and the JOURNAL OF METALS. Mr. Stanley had resigned to join the staff of the Penton Publishing Co.

Mr. Smith has recently been in charge of advertising and sales promotion for the Star-Kimble Motor Div. of the Miehle Printing Press & Mfg. Co. at Bloomfield, N. J. He has a degree of Bachelor of Business Administration from Upsala College, and has done graduate work at Seton Hall University and Rutgers. He is a member of the U. S. Naval Reserve. Mr. Smith, who is 28 years old, lives in East Orange, N. J. with his wife and two sons.

H. DeWITT SMITH

PRESIDENT, AIME, 1955

H. DeWITT SMITH sits behind his desk at 14 Wall St., New York, and talks easily and knowledgeably of many things. The sun streams through the window at his back and plays strange light patterns on the map of Africa on the wall to his left. The conversation ranges from Alaskan mining camps to Mexico, to South Africa, to England, Bolivia, and Chile. Through all the years of his association with mining, his chief interest has been copper. The winning of that metal has brought Mr. Smith into contact with some pretty rugged living. Today he is still active in South African copper mining. He retired as a vice president of Newmont Mining Co. in September 1954, but is still a member of the board, chairman of O'okiep Copper Co. Ltd. and consulting engineer for O'okiep and Tsumeb.

From the time he joined Newmont, Mr. Smith participated in many large copper property merger transactions, including Rhodesian Congo Border Concession Ltd.—Bwana M'Kuba; Phelps Dodge—Calumet & Arizona; and the Phelps Dodge purchase of United Verde. It is with the development of Newmont's South African ventures that Mr. Smith is most closely identified.

As the conversation moves from one mining topic to another, Mr. Smith ties the minerals industry history of the last 50 years into a unified whole. As the story unfolds he pauses for a moment and then ponders that "The romance seems to have gone out of mining for a lot of people." Mr. Smith doesn't agree.

"If there is no longer romance in the minerals industry it is only because no one wants to find it there."

Born Sept. 30, 1888 in Plantsville, Conn., H. DeWitt Smith received his E. M. from Yale's Sheffield School in 1910. His travels started early in his career. As one of the last junior engineers trained by Joshua Edward Spurr, he saw most of the western part of the U. S. and much of Mexico.

Mexico produced several bits of excitement. Mr. Smith was working at the Santa Eulalia mine in 1910 when Mexico erupted in revolution. The rebels swept toward the mine, capturing the town. Federal troops promptly recaptured it. Two years later, while he was working at Sierra Mojada the railroad to Torreon was cut. Mr. Smith got out of the way by pumping a handcar to Chihuahua City.

Speaking of that period of his career, Mr. Smith recalls that "I decided that a geologist must have a basic understanding of mine operations. So when the chance came along to go to Kennecott's Alaskan copper mine as foreman, I took it." The mine crew was one of the best he has ever seen, composed of hardbitten, experienced prospectors, and husky Scandinavians and Russians—a combination of skill and strength, he noted. "They were strong men, not afraid to work." The Alaskan operation proved to be the basis for Kennecott's tremendous growth in the copper business. Before leaving Alaska, Mr. Smith rose to become successively mine superintendent, and assistant manager. He then went to United Verde in Jerome, Ariz., first as mine superintendent and then as general superintendent.

In 1924 Mr. Smith put aside the rough clothes of a mining man and went into the industrial department of the New York Trust Co. "One of the remarkable things about the department at that time, was that the personnel consisted largely of mining engineers." Mr. Smith performed such diverse duties as operating a silk ribbon business and a milking machine company before he left the bank to return to mining and United Verde. Finally, in 1930 he joined Newmont. Since then, there has been only one break in the relationship. Mr. Smith was executive vice president of the Metals Reserve Co., Reconstruction Finance Corp. subsidiary. He was in charge of all domestic metals procurement activities, from 1941 to 1944.

While with Metals Reserve he established a reputation for quickly cutting away red tape and superfluous negotiations on contracts. One representative of a large company walked into the Metals Reserve offices expecting hours of wrangling over vague details. Fifteen minutes later he and his lawyer were walking out the door, the executive recalls, with a signed contract. The soundness of the contracts arranged by Metals Reserve was proven after the end of the war when not one of its agreements had to be renegotiated. An escape clause in each pact elaborated on the exact procedure to be followed for its termination or at the end of hostilities.

In 1939, the British liner Athenia was torpedoed off the Hebrides, with Mr. and Mrs. Smith and one of their daughters aboard. As the passengers were loaded into the lifeboats the family was separated and not until days later did they know of each other's safety. Mr. Smith remembers that he had to bail for dear life to help keep the lifeboat he was in afloat. Eventually, all survivors were picked up and the Smith family was reunited.

Mr. Smith joined the Institute in 1911 and has been active in its affairs for many years, holding office as Chairman of the Nominating Committee in 1939, as a Director, Chairman of the Seeley W. Mudd Memorial Fund Committee, and Vice Chairman of the 75th Anniversary Committee. One of his prime objectives as 1955 President of the AIME will be to strengthen the grass roots movement among the sections that was started during previous presidencies.

Mr. Smith is vitally concerned with the problem of attracting young men to the mining industry, and has been instrumental in the O'okiep program which brings young men to the U. S. from South Africa each year. The South Africans study in U. S. schools for three years under the plan. Many of his training ideas evolved from the United Verde mining training program that operated in the years following World War I. Mr. Smith feels that the training methods used were some of the best ever practiced in the domestic mining industry. Mining engineers who came under the United Verde influence at that time have moved on to positions of leadership throughout the industry.

Mr. Smith believes that the best way to start in the mining business is as a mucker, miner, or timberman. Another way to attract the kind of men the mining industry needs is the payment of adequate wages. In the final analysis it's up to management to provide the climate that will make mining attractive.

Mr. Smith married Ellen Dawson Burke of Plainfield, N. J. in 1916. They have three children, C. DeWitt Smith, Mrs. Alfred Sanford, II, and Mrs. J. Clifton Rodes. C. DeWitt Smith is with St. Joseph Lead Co., carrying on the family tradition of making one's own way.

If there is one thing in the world that will distract Mr. Smith from mining and the romance of copper, it's surfcasting. Nantucket beaches, early in the morning or just before sundown, are his favorite spots. His biggest catches have been a 9½ lb bluefish and a 23½ lb striped bass, both hooked and landed off Nantucket.

Around the Sections

• Dean M. P. O'Brien, College of Engineering, University of California, spoke on *Modern Educational Trends in Engineering* at a recent meeting of the **San Francisco Section.** The get-together was held at the Shattuck Hotel with Shannon Fowler, president of the Mineral Technology Assn., (the U.C. student section), presiding. At another gathering, the Annual Reminiscence Meeting, Ken Karch, noted comedian, played master of ceremonies. The program included five acts from the best floor shows in San Francisco. At last month's meeting, Granville S. Borden, tax counsel for the Standard Oil Co. of California and vice president and director of Idaho Maryland Mines Corp., spoke on *Taxation and the Mineral Industry.*

• Annual business meeting of the **Colorado Section** took the form of a dinner held at the University Club, Denver, with Ben H. Parker, Section Chairman, presiding. Some 121 persons, including members and guests attended. Speaker was Sheldon P. Wimpfen, manager, Grand Junction Operations Office, AEC. Officers elected for 1955 are: C. L. Barker, Chairman; R. L. Scott, Vice Chairman; J. V. Thompson, Secretary-Treasurer; and Ben H. Parker, W. H. Burgin, L. J. Parkinson, Directors, and T. C. Heistand, Director representing Petroleum Section. Mr. Wimpfen spoke on *Developments in Uranium Exploration on the Colorado Plateau and the Peacetime Future of the Industry.*

• Institute President Leo F. Reinartz visited the **Washington, D. C. Section** last month. The meeting was held at the National Press Club, with Mr. Reinartz discussing the *State of the Institute.* During his talk he made note of the fact that E. D. Gardner, section member, is to receive the Daniel C. Jackling Award.

• The **St. Louis Section,** at a recent gathering, heard LeRoy Scharon, professor of geophysics, Washington University, discuss the geology, exploration, mining, milling, and metallurgical aspects of uranium in relation to the Colorado Plateau. The dinner-meeting was held at Washington University.

• Last month the **New York Section,** with the invaluable cooperation of the Ladies Auxiliary, held a reunion meeting attended by miners, metallurgists, and petroleum men, along with their wives, in the Palm Terrace Suite of the Hotel Roosevelt. One of the features of the evening was the absence of speeches and the high quality of the entertainment provided by the section.

• The **Montana Section** held its annual election dinner in the Silver Bow Room of the Finlen Hotel. The following officers were elected for the coming year: Clifford P. Milkwick, Chairman; J. Robert Van Pelt, Vice Chairman; Stanley M. Lane, Executive Committeeman; Thomas K. Graham and Fred Owsley, Committeemen; and Clifford Hicks, Secretary-Treasurer.

• The **Chicago Section** meeting in January featured Walter H. Mulflur, manager of hot and cold strip mills, Algoma Steel Co., speaking on *Operations of a Combined Bar and Cold Strip Mill.* Meeting was held at the Chicago Bar Assn. This month, the Chicago group presents *Oxygen Steel Making in Canada,* by F. J. McMulkin, research and development engineer, Dominion Foundries & Steel Ltd.

• James W. Keim has been elected Chairman of the **Bisbee-Douglas Subsection** for 1955. Arthur F. Himebaugh, Vice Chairman; Robert C. Meyer, Secretary-Treasurer, and Joseph Kolessar, Jr., Membership Chairman, complete the slate. F. W. Galbraith of the University of Arizona presented an illustrated talk on *Prospecting in the Yukon.*

• Bob Erskine, engineer, EMSCO Mfg. Co., spoke on *Dual Zone Pumping Installations* at the last meeting of the **Kansas Section.** The meeting took place at the Oriental Restaurant, Wichita.

A challenging and dramatic film on highway safety, *The Perfect Crime,* has just been released by Caterpillar Tractor Co., Peoria, Ill., through its local dealers. The 20½ min sound-color production is offered by Caterpillar with the cooperation of the National Safety Council's Construction Section and members of the Associated General Contractors of America. The movie opens up with a double murder arising from a $14 robbery. Public indignation is aroused and the murderer is caught quickly. The narrator then draws a comparison with another crime—murder on the highways—and cites an apathetic public for its indifference to this tragedy.

• U. S. Steel Corp. has just published the 14th edition of a catalog describing educational and entertaining industrial motion pictures sponsored and distributed by the firm. All U. S. Steel films are available on free loan basis. Write U. S. Steel Corp., 525 William Penn Place, Pittsburgh 30.

Personals

Joseph C. Mead is supervisor of quality control dept., National Lead Co. of Ohio, at Cincinnati.

D. H. Lillicrap has returned to England after completing his contract with ACSM Champion Reef Oorgaum, Kolar Gold Field, Mysore State, India.

Raymond E. Lamborn, assistant geologist, Div. of Geological Survey, Ohio State University, Columbus, is the author of a recently published report, *Geology of Coshocton County.* (See books page 108.)

Richard A. Mullins has been named preparation manager, Enos Coal Mining Co., Oakland City, Ind., to succeed the late **William Caler.** Mr. Mullins will also be preparation manager for Enoco Collieries Inc., Vincennes, Ind. He was formerly chief chemist, Ayrshire Collieries Corp., Danville, Ill.

Luis A. Nogales, after resigning from the Bolivian Mining Corp. where he was manager of production, is now working as consulting mining engineer, Casilla 1106, La Paz, Bolivia.

John F. Dugan, general superintendent of mines, International Smelting & Refining Co., a subsidiary of Anaconda Copper Mining Co., Salt Lake City, has retired, but will continue as consultant. Mr. Dugan was graduated from Montana School of Mines in 1906 and joined Anaconda the same year.

Annan Cook, formerly district geologist, Eastern District, Bear Creek Mining Co., has been since July 1 district geologist in charge of the southwestern U. S. for Bear Creek. Mr. Cook's headquarters are in the Hobart Bldg., San Francisco.

Joseph Daniels, professor emeritus of mining engineering and metallurgy, University of Washington, is a member of Washington State College educational mission to Pakistan. Mr. Daniels will establish a department of mineral engineering at Punjab University College of Engineering and Technology in Lahore. During this two-year mission his principal duties will be to organize a curriculum, but he will also study Pakistan's mineral resources.

F. W. Libbey, director of Oregon State dept. of geology and mineral industries, Portland, was retired November 1 and **Hollis M. Dole,** dept. geologist, who has been Mr. Libbey's assistant since September, has taken over as acting director.

Dan McCutchen of Beckley, W. Va., graduate of Colorado School of Mines, is now a sales representative for Atlas Powder Co.

Charles Camsell was recently elected an Honorary Life Member of the Canadian Geographical Society, an organization of which he was one of the founders and later president for 11 years. His recent book, *Son of the North,* is a human interest story of his early life in the Mackenzie River country and of his explorations in that region.

Felix E. Wormser, Asst. Secretary of the Interior for Mineral Resources, with U. S. Bureau of Mines employees who received Distinguished Service Awards at the 14th Honor Awards Convocation of the U. S. Dept. of the Interior, Washington, D. C., on Dec. 14, 1954. Left to right: John S. Ball, Walter I. R. Murphy, Stephen M. Shelton, Assistant Secretary Wormser, Vernon F. Parry, and Harold S. Kennedy. Not in the group is Richard D. Leitch, who retired from the USBM last spring and received his Distinguished Service Award in absentia.

T. B. COUNSELMAN

T. B. Counselman, vice president, AIME, has left The Dorr Co. where he was manager, FluoSolids Div., to join Behre Dolbear & Co., New York mineral consultants in mining, geology, metallurgy, and management. A graduate of Columbia University, Mr. Counselman gained his early experience in both mining and ore dressing with Inspiration Copper Co., Cananea Consolidated Copper Co., Arizona Copper Co., and Miami Copper Co. He worked with **Daniel C. Jackling** on the first of the taconite ventures, the Mesabi Iron Co. Mr. Counselman joined The Dorr Co. in 1927. His inventions include magnetic separators, classifiers, and equipment for manufacturing synthetic rubber. He is the author of numerous technical articles on concentration of low grade iron ores, cement manufacture, and handling flue dust in blast furnace plants.

John Herbert Hollomon, manager, metallurgy dept., General Electric Research Laboratory, Schenectady, N. Y., was honored January 22 at Louisville, Ky., by the National Junior Chamber of Commerce as one of America's ten outstanding young men of 1954. Mr. Hollomon was given the award for his leadership in metallurgy and metallurgical research, and for special service to his country in war and peace.

Grover C. Miller, geologist, Tennessee Coal & Iron Div., U. S. Steel, retired Oct. 1, 1954 after 40 years of service. His successor is **Donald Carnes** whose office is in Fairfield, Ala.

Neil F. Ritchey has been appointed to the new post of director of nuclear equipment and construction, Knapp Mills Inc., New York. Mr. Ritchey, formerly an atomic energy advisor for Reynolds Metals Co., Louisville, Ky., is a graduate of the University of Indiana and the Advanced School of Reactor Technology, Oak Ridge, Tenn.

John D. Vincent, who was with Northern Peru Mining & Smelting Co., Tacna, Peru, is now with American Smelting & Refining Co., Salt Lake City.

PHILLIP L. MERRITT

Phillip L. Merritt, assistant director for exploration, AEC Div. of Raw Materials, Washington, D. C., has resigned to accept a position as senior geologist with E. J. Longyear & Co. of Minneapolis. He will maintain offices in the Graybar Bldg. in New York. **Robert D. Nininger,** who has been deputy assistant director for exploration, is acting director in Mr. Merritt's place. Mr. Merritt's experience as a geologist dates from 1929, when he conducted geological work in Northern Rhodesia and South Africa. From 1934 to 1936 he was a geologist in the Dept. of Mines, Colombia. In 1936 he joined American Cyanamid Co., where he was until he joined the AEC in 1946.

Glenn E. Sorenson, president of Kemmerer Coal Co., Kemmerer, Wyo., and an officer in several affiliates, was elected president of the Gunn-Quealy Coal Co. at a meeting of directors in New York.

C. A. Weekley is spending several weeks in Japan assisting in the negotiations for a $2.5 million loan from the Japanese steel mills for a beneficiation plant at Larap, P. I., for the Philippine Iron Mines. Much of the equipment will be of Japanese manufacture, but largely under license agreement from U. S. manufacturers, such as Allis-Chalmers, Nordberg, and Goodrich Rubber Co. Mr. Weekley will supervise completion of plant design and construction. He also designed and constructed the 4000-ton copper flotation plant at Toledo, Cebu.

Paul R. Geoffroy, consulting engineer, Val d'Or, Que., was recently elected president of Vendome Mines Ltd., president of Belfort Mines Ltd., and vice president of Luteca Development Ltd. These three companies are involved in mining developments in the Province of Quebec with headquarters in Montreal.

Merle H. Guise is now in San Diego and Imperial counties prospecting for uranium, tantalite-columbite, and rare earths. He was in Nevada and California exploring for tungsten, mercury, and uranium, and spent the summer exploring on the Plateau.

LOUIS A. PANEK

Louis A. Panek has been selected as the Warren Lecturer at the Minnesota School of Mines and Metallurgy for 1955. This lectureship was established in memory of George H. Warren, pioneer educator and lumberman in Minnesota, and his son, Frank M. Warren, well-known mining engineer. Its purpose is to promote effective teaching and education in the mineral industry field by bringing specialists and men of outstanding ability to lecture at the university. Mr. Panek, on leave of absence from the Applied Physics Branch, U. S. Bureau of Mines, College Park, Md., is to give a series of about 30 lectures on rock mechanics and the design of mine openings.

RALPH E. KIRK

Ralph E. Kirk, AIME Director and formerly manager of raw materials, Tennessee Coal & Iron Div., U. S. Steel Corp., Fairfield, Ala, and **Percy G. Cowin** have formed the new consulting engineering firm of Kirk & Cowin, Birmingham. A graduate of Pennsylvania State University, Mr. Kirk was superintendent of Luzerne Coal & Coke Co. from 1914 to 1916. In 1916 he first became associated with U. S. Steel when he accepted the position of assistant superintendent, Colonial Three and Four and Bridgeport mines of H. C. Frick Coke Co., Scottdale, Pa. Mr. Kirk volunteered for duty with the U. S. Army during World War I and after his discharge with the rank of captain, rejoined the Frick Coke Co. In 1930 he became general superintendent, Mahanoy Div., Philadelphia & Reading Coal & Iron Co. and in 1936, Mr. Kirk went to Alabama to resume his association with U. S. Steel as general superintendent of coal mines for TCI. Under his direction TCI's coal mines were extensively mechanized, modernized, and developed to a position of leadership in the bituminous-coal industry. Among other affiliations, Mr. Kirk is a member of the Old Timers Club of the American Mining Congress, the Coal Mining Institute of America, and the American Zinc Institute. He was the 1953 national chairman of the AIME Coal Div. and the 1954 chairman of the Mining Branch Council. Cowin & Co. will continue as a mine plant and shaft sinking contractor, with Mr. Cowin as president of that company and of the Mine & Contractors Supply Co., also located in Birmingham.

David W. Chase, who was with Cerro de Pasco Corp., Mahr Tunel, Peru, is now with Northern Peru Mining & Smelting Co., Quiruvilca Unit, Trujillo, Peru.

George S. Ryan, formerly assistant geologist, Anaconda Copper Mining Co., Reno, Nev., is now a second lieutenant in the Corps of Engineers assigned to the 517th Engineering Terrain Intelligence Detachment.

Gordon K. Teal, assistant vice president, has been appointed to head the Research Div., Texas Instruments Inc. As assistant vice president, research for the Dallas-based electronics and geophysics firm, Mr. Teal will have charge of all TI research, involving work in many phases of electronics and geophysics. Mr. Teal previously headed the materials and components research dept., with primary responsibility for semiconductor research. **R. W. Olson,** TI vice president formerly in charge of research and engineering, in 1954 assumed the presidency of Houston Technical Laboratories, wholly owned TI subsidiary operating as the Petroleum Instrumentation Div. Mr. Olson has moved to Houston to head the expanding Houston operation. All TI geophysical instrumentation activities have been transferred to HTL and a 6-acre tract has been purchased in Houston for new plant construction.

Frank M. Estes has left the U. S. on a professional trip to Colombia. He will be gone for several months.

John Louis Dupont, engineer with Isbell Construction Co., Silverbell, Ariz., for the past two years, is now in charge of this company's Phoenix area. Mr. Dupont is working at five open mines, two in Arizona and three in New Mexico, and doing highway and mine estimating for bid purposes.

OTTO HERRES

Otto Herres, vice president, Combined Metals Reduction Co., has been elected president for 1955 of the Utah Mining Assn., Kearns Bldg., Salt Lake City. **L. F. Pett,** general manager, Utah Copper Div., Kennecott Copper Corp., was elected first vice president and **Clark L. Wilson,** vice president and manager of operations, New Park Mining Co., was elected second vice president. **A. G. Mackenzie** is vice president and consultant, **Miles P. Romney,** secretary-manager, and **Walter M. Horne,** assistant secretary-manager.

P. W. Richardson has joined the Benguet Consolidated Mining Co., Baguio, P. I., as a mine foreman.

Obituaries

Charles M. Brown (Member 1945) died Aug. 10, 1954. He was president and treasurer, Brown Ice & Coal Co., Anderson, S. C. Mr. Brown was born in 1886 in Belton, S. C., and studied at Georgia School of Technology. He went to Mexico in 1904 and first worked at the San Juan mine, Ocotlan, Oaxaca. Among other companies, Mr. Brown was superintendent, Colonial Mines Co., Sierra, Juarez, and superintendent, Barron mine, Cia. Real del Monte, Pachuca. He also prospected for several years in southwest Mexico and then was general foreman of mines for Cerro de Pasco Corp. in Peru. In 1919 Mr. Brown became a partner in Brown & West, Belton, S. C., and from 1927 to 1932 he was superintendent, Colonial Ice Co., Anderson, S. C.

James J. Carrigan (Member 1917), retired manager of mines, Anaconda Copper Mining Co., Butte, Mont., died suddenly Oct. 17, 1954. He was born in 1886 in Hancock, Mich., and was graduated from Michigan College of Mines in 1908. Mr. Carrigan took his first job with Anaconda as a miner in the Gagnon and later transferred to the St. Lawrence mine at Butte as an underground worker. In 1910 he was employed by the geological dept. and remained there until 1912. His next assignment was as a shift boss of the Badger mine. Early in 1914 he was appointed safety engineer for Anaconda and the following year became superintendent of zinc mines. He was named chief assistant general superintendent of mines in 1926 and became general superintendent of mines in 1930. In 1940 he was appointed manager of mines, a position he held until 1947 when he retired due to ill health. For 40 years Mr. Carrigan was a popular resident of Butte and outstanding in mining circles all over the world. He was past chairman of the Montana Section, AIME, and the Montana Society of Engineers. Since his retirement he had spent part of each year in Montana and part in California.

George A. Guess (Member 1905) died Oct. 21, 1954. He was professor emeritus of metallurgical engineering, University of Toronto. Mr. Guess was born in 1873 at Hartington, Ont., and received his M.A. in 1894 from Queen's University, Kingston, Canada. After working with mining and smelting companies in British Columbia, he served as chief metallurgist with Tennessee Copper Co., Copper Hill, Tenn., and with the Cananea Copper Co. in Mexico. He then went to Peru as metallurgical superintendent for Cerro de Pasco Corp.

In 1914 Mr. Guess joined the faculty of the University of Toronto and organized the metallurgical engineering dept., which he directed until 1940 when he retired. A specialist in copper metallurgy, Mr. Guess conducted experiments for the Dominion Government during World War I to determine the possibility of recovering iron from the slag of the smelters in the Sudbury area. He later served on the Ontario Government's iron commission. Mr. Guess wrote many articles for technical magazines and was also the author of a work on philosophy. He was a member of the Canadian Institute of Mining and Metallurgy and the Engineers' Club of Toronto.

Frederick John Kasper (Member 1944) died Oct. 19, 1954. He was a combustion engineer with Eastern Gas & Fuel Associates. Mr. Kasper was born in Boston in 1890 and studied nights at Lowell Institute and Wentworth Institute while working for the Fuel Testing Co. in Boston. He was with the U. S. Army Engineers during World War I and afterwards was employed by Perry Barker Engineering Co. in Boston. Before joining Koppers Coal Div., Mr. Kasper was with Coleman & Co. Inc. in Philadelphia for more than eight years. During World War II he served with the Solid Fuels Administration as a consultant on problems of coal conservation.

Mack C. Lake (Member 1921) died Nov. 9, 1954 in San Francisco following a lengthy illness. He was a former president of Orinoco Mining Co., a U.S. Steel Corp. subsidiary. Mr. Lake retired in January 1954, but was retained by the corporation as a consultant on special assignments in the engineering and raw materials dept. He was born in Brodhead, Wis., in 1890 and received his B.S. in 1914 from the University of Wisconsin. Following graduation, Mr. Lake took a year of postgraduate work in geology at Wisconsin and assisted C. K. Leith, professor of geology, in various geological investigations. In 1915 he joined the M. A. Hanna Co., Duluth, as geologist in connection with its operations in the Lake Superior district. Mr. Lake joined U.S. Steel in 1945. Exploration under his direction led to the discovery in 1947 of the iron ore deposits at Cerro Bolivar in the remote interior of Venezuela. Mr. Lake lived to see the completion of the Cerro Bolivar development and the shipment of Venezuelan ore to U.S. steel mills. Among other companies he was at various times vice president and director, Chapin Exploration Co., Arisota Corp., Manganese Ore Co., Hanna Development Co., and president and director, Calmich Mining Co. Mr. Lake contributed articles to technical magazines and was a member of the Society of Economic Geologists, the Mining Club of New York, and the Engineers Club of San Francisco.

Edmund J. Longyear died Dec. 4, 1954. He played a predominant part in the early development of Minnesota's Mesabi Range iron ores and superintended the first diamond drilling on the range in 1890. He was also founder of E. J. Longyear Co. of Minneapolis, which has built up a world-wide business.

Born in Grass Lake, Mich., in 1864, Mr. Longyear attended the University of Michigan and Michigan College of Mines. After a job on a railway survey he went to work for his cousin, J. M. Longyear of Marquette, Mich., and had his first contact with diamond drilling on a job on the Menominee Range of Michigan.

In 1890 J. M. Longyear asked E. J. Longyear to take charge of exploration in northern Minnesota and for the next 35 years he was closely connected with the exploration and development of the range. On June 3, 1890 he started the first diamond drill hole on the range at a site 74 miles north of Two Harbors, and in the fall of that year brought in a second diamond drill.

From his initial work for his cousin, J. M. Longyear, E. J. Longyear branched into contract drilling for others. Work with the Mallman Iron Co. was followed by extensive drilling for Longyear-Mesabi Land & Iron Co., the Pillsbury, Bennett and Longyear group, and the Wright-Davis Logging Co. Mr. Longyear built his first permanent office in Hibbing, Minn., in 1896 and took an active part in the life of the growing village.

As exploration drilling on the Mesabi declined, the Longyear operations were extended to other parts of the country. He was asked by Colonel Greenway to send a drill to Arizona in 1914 and soon after had six diamond drills in operation on what is now the New Cornelia mine of the Phelps Dodge Copper Corp. This was the first successful application of diamond drilling to a low grade copper deposit, and subsequent comparison of mine samples with the diamond drill hole samples showed a close check on copper content.

With the growth of contract activities outside of Minnesota, Mr. Longyear's Hibbing organization and his Marquette, Mich., organization, Longyear & Hodge, were consolidated under the name of E. J. Longyear Co. with headquarters at Minneapolis.

First important operation of the Shaft-Sinking Div. was an 1100-ft shaft near Lokkens Verk, Norway, for Orkla Grube Aktiebolag. This was started in May 1915 and successfully completed in the fall of 1916. Some of the returning personnel had harrowing experiences dodging German U-boats then at the height of their activity during the middle part of World War I.

Foreign activities of the Contract Drilling Div. started with contracts in Cuba in 1912 to 1916, and the first large operation was a six-drill con-

Necrology

Date Elected	Name	Date of Death
1946	Peter M. Anderson	Nov. 5, 1954
1952	J. R. Crenshaw	Unknown
1942	Michael Dwyer	Dec. 1953
1915	L. B. Eames	Dec. 3, 1953
1938	Alfred H. Geisler	Dec. 7, 1954
1945	Ray G. Gibson	Oct. 16, 1954
1923	Silas L. Gillan	Nov. 9, 1954
1947	Francis G. Hoffman	Nov. 24, 1954
1916	William Wallace Inglis	Jan. 19, 1953
1937	C. H. McNaughton	Unknown
1914	Royden C. Philpott	Feb. 7, 1954
1929	Erwin Sohn	Unknown
1946	E. J. Tompkins	Aug. 25, 1954
1921	Jay Tuttle	Nov. 3, 1954
1945	Lawrence Vander Leck	Unknown
1921	Alexander Walker	Aug. 3, 1954
1944	Everett Wilcox	Nov. 2, 1954
1953	John E. Wilcox	Unknown

tract carried out in Yunnan, China, in 1919 to 1920. Drilling equipment was taken by boat up the Irrawaddy River from Rangoon, Burma, to Mandalay. From there it was carried by a 500-man coolie train over a trail, subsequently part of the Burma Road of World War II. During the 1920's various foreign contracts were carried out in Mexico, Cuba, and South America. The next large operation was a 9-drill contract carried out in Northern Rhodesia in 1929 to 1930 for Roan Antelope and Mufulira Copper Mines Ltd., subsidiaries of Rhodesian Selection Trust of London. Recently the division completed a 7-drill contract with Oliver Iron Mining Co. on the much-publicized Cerro Bolivar iron deposit in northeastern Venezuela.

Manufacturing activities of the company grew from a small diamond drill repair shop started by Longyear & Hodge in Marquette, Mich., about 1902, until today the company is second largest U. S. manufacturer of diamond drills.

Mr. Longyear continued as active head of the company until he moved to California in 1924, when he was succeeded as president by his son, Robert. He continued, however, as vice president and director. Mr. Longyear will be remembered by those who knew him as a man of highest professional standards and integrity.

Winston Walter Spencer

An Appreciation by
C. M. Romanowitz

An accident at Platinum, Alaska, on Sept. 21, 1954 caused the untimely death of Winston Walter Spencer (Member 1935). He was chief engineer for Goodnews Bay Mining Co. at Platinum and had been associated with that company since 1936. His wife, Renée, survives him and to her are extended the condolences of many friends in mining and dredging.

Born in Craig, Colo., on Feb. 16, 1905, Winston went to Alaska in 1926, first worked with the U. S. Public Land Office in Southeastern Alaska, and then for two years was associated with the McRae-Patty mining interests in the Fairbanks area.

He was graduated from the University of Alaska in 1934 with a B.S. degree in geology and mining, and in 1949 received the degree of mining engineer from the same university.

His work with the Goodnews Bay Mining Co. has been outstanding and his skill and knowledge of dredging methods were responsible, in a large measure, for the continued success of the mining at Goodnews Bay.

A large part of Winston's success in his chosen career was brought about by his friendly nature and willingness to work with his associates in developing any new ideas that might be advanced by any member of the organization. Few engineers have the broad viewpoint toward life that he had and which made it possible for him to secure the cooperation of every one who worked with and for him. His contributions to the mining engineering profession will serve as a memorial to his professional skill. His warm spirit of friendliness will remain emplanted in the memories of his many friends and the associates with whom he worked.

Swain Joseph Swainson

An Appreciation by
Theodore B. Counselman

Swain Joseph Swainson, Member of 1934, died Friday, Oct. 22 at Southold, Long Island after a prolonged illness. He was buried at Westfield, N. J.

He was born in 1901 at Mountain, N. D. and attended the University of North Dakota, from which he received a B.S. degree in 1923, and the University of Utah, from which he graduated with an M.S. degree in 1925. He first did some research work in ore dressing at the University of Utah, and joined the staff of American Cyanamid as a metallurgist in 1926.

He had much to do with the establishment of that company's mineral dressing laboratories at Stamford, Conn., which are among the largest and most extensive in the world, and for many years was director. His major field of work was in the beneficiation of ores and the extraction of their minerals. He was one of the first to appreciate the importance of the microscope in the solution of mineral dressing problems, and used this technique extensively. He invented many processes related to this field and was the author of numerous papers which were published in technical journals. He was particularly noted for his work in improvements in froth flotation, and in Heavy-Media separations.

For several years Mr. Swainson supervised the operations of the Winchester research laboratory of the Atomic Energy Commission, operated at that time under contract by American Cyanamid. This work was concerned with the technique of extraction of uranium from its ores.

Mr. Swainson was very active in various technical societies, particularly in the American Institute of Mining and Metallurgical Engineers. He was on several committees of the Minerals Beneficiation Div., after helping to form that division, and finally became its Chairman and therefore a member of the very exclusive Molahominem Bellum Distinctionus (Mill Gentlemen of Distinction). He was a member of the Robert H. Richards Award Committee and was Chairman at the time of his death. He was also a member of the Mining and Metallurgical Society of America, and the Chemical, Metallurgical and Mining Society of South Africa. Besides belonging to the Mining Club of New York, he was a member of Sigma Alpha Epsilon and of the honorary Scientific fraternity, Sigma Xi.

Mr. Swainson leaves his wife, Cary Mulford Swainson, his father, Oddur Swainson, a sister, Bertha Swainson, and a host of warm friends throughout the profession.

Proposed for Membership

MINING BRANCH, AIME

Total AIME membership on Dec. 31, 1954 was 21,816; in addition 2179 Student Associates were enrolled.

ADMISSIONS COMMITTEE

O. B. J. Fraser, Chairman; R. B. Caples, Vice-Chairman; F. A. Ayer, A. C. Brinker, R. H. Dickson, Max Gensamer, Ivan A. Given, Fred W. Hanson, T. D. Jones, Sidney Rolle, J. H. Scaff, John T. Sherman, F. T. Sisco, Frank T. Weems, R. L. Ziegfeld.

The Institute desires to extend its privileges to every person to whom it can be of service, but does not desire as members persons who are unqualified. Institute members are urged to review this list as soon as possible and immediately to inform the Secretary's office if names of people are found who are known to be unqualified for AIME membership.

In the following list C/S means change of status: R, reinstatement; M, Member; J, Junior Member; A, Associate Member; S, Student Associate.

Alabama
Birmingham—Gibb, Erskine G. (M)
Birmingham—Marshall, Alexander J., Jr. (A)
Birmingham—Postell, George W., Jr. (M)
Hueytown—McCullar, Barney S. (M)

Arizona
Ajo—Gensman, Owen A. (C/S—A-M)
Douglas—Allen, William, Jr. (R. C/S—J-M)
Douglas—Ludden, Raymond W., Jr. (R. C/S—S-M)
Inspiration—Power, Kenneth L. (R. C/S—S-M)
Morenci—Bacon, Vance N. (J)
Morenci—O'Neill, George H. (J)
Phoenix—Gentry, Jack T. (R. C/S—S-A)
Phoenix—Haentjens, Richard P. (A)
San Manuel—Blucher, Arthur G., Jr. (J)

California
Arlington—Tillson, Alexandra (M)
Atherton—Christensen, Allen D. (M)
Lafayette—Lindsay, Adrian K. (R. C/S—S-M)
Matheson—Dubach, Merrill Leon, Jr. (J)
Oakland—Charles, Abbott (C/S—A-M)
Oakland—Weishaar, Gerard P. (A)
San Francisco—Bergis, Robert R. (R. C/S—J-M)
San Francisco—MonteVerda, Bernard (M)
Walteria—Stone, William D. (R. M)

Colorado
Denver—Bhappu, Roshan B. (R. C/S—S-J)
Golden—Hodder, Edwin J., Jr. (J)
Grand Junction—Appelin, Carl W. (C/S—A-M)
Grand Junction—Burnett, John T. (R. C/S—S-M)
Grand Junction—Hill, John F. (R. C/S—S-M)
Grand Junction—Hollis, Roy F. (R. M)

Idaho
Montpelier—Wright, John S. (R. C/S—S-M)
Mullan—Love, William H. (C/S—A-M)
Wallace—McMahon, Kenneth H. (J)

Illinois
Rosiclare—Skeels, Walter S. (R. C/S—J-M)
St. Charles—Stevens, Arthur J. (J)

Massachusetts
Cambridge—Jacobson, Herbert S. (J)
Swampscott—Breymann, John B., III (C/S—A-M)

Michigan
East Lansing—Wesner, Richard F. (C/S—J-M)
Iron River—Dewald, Walter E. (M)
Ishpeming—Brown, Douglas E. (R. C/S—S-J)

Minnesota
Crosby—Simmons, John J., Jr. (J)
Crosby—Walker, Neal E. (J)
Deerwood—La Sart, Elton J. (M)
Duluth—Alworth, Royal D., Jr. (M)
Minneapolis—Moerlein, George A. (J)

Montana
Butte—Wiseman, Blaine L. (R. C/S—S-J)

Nevada
Boulder City—Fischer, Francois A. (M)
Boulder City—Laughton, George L. (A)
Boulder City—West, John M. (J)
Red House—Hardy, Royce A. (C/S—A-M)

New Jersey
Franklin—Cavanaugh, John P. (J)
Metuchen—Anderson, William W., Jr. (J)
New Brunswick—Chang, Chen-Siang (C/S—A-M)
Plainfield—Jessen, Heinrich C. (R. M)
Somerville—Streib, William C. (M)

New Mexico
Hurley—Brubaker, Joseph J. (M)
Tyrone—Finton, Murrel H. (M)

New York
Buffalo—Zinter, Theodore C. (M)
New York—Campbell, Douglas S. (J)
Rochester—Brown, David L. (R. J)
Troy—McCarthy, Peter (A)

Ohio
Cincinnati—Batchelor, William C. (A)
Columbus—Patton, Robert M. (M)

Oklahoma
Miami—Elizondo, Joseph B. (C/S—A-M)

Pennsylvania
Fairchance—Peters, Franklin W. (M)
Johnstown—Rickard, Rembrandt P. (A)
Johnstown—Slick, Edwin E. (A)
Kittanning—Stevens, Donald R., Jr. (A)
Pittsburgh—Craig, John D., Jr. (A)
Uniontown—Johnson, Henry R. (A)

South Carolina
Travelers Rest—Stewart, Otis F. (M)

Texas
El Paso—Bews, Alastair I. (A)
Houston—Lloyd, C. Leonard (M)

Utah
Moab—Ball, Lauren L. (J)
Moab—Steen, Charles A. (R. C/S—J-M)
Salt Lake City—Anderson, Donald O. (A)
Salt Lake City—Burwell, Blair T. (M)
Salt Lake City—Cox, Fred E. (A)
Salt Lake City—Torgerson, Earl O. (J)

Vermont
South Strafford—Dawe, George L. (J)
South Strafford—Millett, Sewell (J)

Virginia
Gate City—Evans, Thaddeus B. (M)
Lynchburg—Stanley, Alan (M)

Washington
Northport—Paparich, Frank M. (A)

West Virginia
Charleston—Lindsay, Kenneth E. (M)

Wisconsin
Milwaukee—Briber, Frank E., Jr. (C/S—A-M)
Milwaukee—Pullen, Fletcher W. (A)
Milwaukee—Wallace, William M. (R. M)
Platteville—Christiansen, Carl R. (C/S—A-M)

Wyoming
Hanna—Patton, Carmal (M)

Bolivia
Llallagua—Echalar, Alberto V. (M)

Canada
N. Ontario, Virginiatown—Martin, Michael D. (J)

Cuba
Oriente—Wandke, Alfred D. (C/S—A-M)

French Morocco
Zellidja Bou Beker—Young, Francis J. M. (M)

French West Africa
Conakry, Fr. Guinea—Burki, Axel N. (J)

Israel
Haifa—Ishai, Shimeon (M)

Mexico
Chihuahua—Brady, Maurice H. (C/S—A-M)

Nicaragua
Managua—Kupsch, Robert J. (R. C/S—S-J)

Northern Rhodesia
Kitwe—Gray, Roy F. (C/S—A-M)

Peru
Arequipa—Kimball, Richard H. (J)
Cerro de Pasco—Saunders, Bill G. (C/S—A-M)
Lima—Palacios, Virgilio (J)

Transvaal, South Africa
P. O. Coalville—Lee, Richard A. (M)

Venezuela
Los Chaguaramos y Los Mangos—v. Kanicky, Leopold (M)

Coming Events

Feb. 14-17, AIME, Annual Meeting, Conrad Hilton Hotel, Chicago.

Feb. 21-24, American Concrete Institute, 51st annual convention, Hotel Schroeder, Milwaukee.

Feb. 23, AIME, Utah Section, University of Utah Debate Teams, dinner meeting with Auxiliary invited to participate, Salt Lake City.

Mar. 1-2, American Society of Mechanical Engineers, First International Congress on Air Pollution, Hotel Statler, New York.

Mar. 9, AIME, Connecticut Local Section, Bridgeport, Conn.

Mar. 10, AIME, Cleveland Local Section, American Room, Manger Hotel, Cleveland.

Mar. 16, AIME, Connecticut Local Section, annual meeting, Statler Hotel, Hartford, Conn.

March 17, AIME, Utah Section, Aims, Functions, and Organization of the USBM, panel discussion, Salt Lake City.

Mar. 20-23, American Institute of Chemical Engineers, Kentucky Hotel, Louisville.

Mar. 28-Apr. 1, Ninth Western Metal Exposition, Pan-Pacific Auditorium, and **Ninth Western Metal Congress,** Ambassador Hotel, Los Angeles.

April 9, AIME, Utah Section, Spring Dinner Dance, Salt Lake City.

Apr. 13, Material Handling Institute, spring meeting 10:00 am, Drake Hotel, Chicago.

Apr. 14, AIME, Cleveland Section, joint meeting with American Ceramic Society, NACA Laboratories, Cleveland Hopkins Airport.

Apr. 18-19, Third National Air Pollution Symposium, Pasadena, Calif.

Apr. 18-20, AIME, Blast Furnace, Coke Oven and Raw Materials, and National Open Hearth Steel Conferences, Bellevue-Stratford Hotel, Philadelphia.

Apr. 18-22, American Society of Mechanical Engineers, Diamond Jubilee spring meeting, Lord Baltimore Hotel, Baltimore.

Apr. 19-21, Canadian Institute of Mining and Metallurgy, annual meeting, Royal York Hotel, Toronto.

April 21, AIME, Utah Section, How the Oil Industry Operates, panel discussion, Salt Lake City.

Apr. 22-23, AIME, New England regional conference, Boston.

Apr. 28-30, AIME, Pacific Northwest Conference, Spokane.

May 16-19, American Mining Congress, 1955 Coal Show, Cleveland.

May 19, AIME, Utah Section, joint meeting with University of Utah Student Chapter. Speaker: Eugene Callaghan, director, New Mexico Bureau of Mines & Mineral Resources.

June 1-18, Joint Metallurgical Societies, European meeting.

June 20-24, American Society of Engineering Education, 63rd annual meeting, Pennsylvania State University, State College, Pa.

June 26-July 1, American Society for Testing Materials, annual meeting, Chalfonte-Haddon Hall, Atlantic City, N. J.

Sept. 19-22, AIME, Industrial Minerals Div., fall meeting, Asheville, N. C.

Oct. 3-5, AIME MGGD fall meeting and Black Hills regional meeting of the Ind. Min. Div., Rapid City, S. D.

Oct. 5-8, AIME, Minerals Beneficiation Div., fall meeting, Salt Lake City.

Oct. 6-8, AIME, Utah Section, Rocky Mountain Industrial Minerals Conference, Salt Lake City.

Oct. 9-13, American Mining Congress, Metal Mining—Industrial Minerals Convention, Las Vegas, Nev.

Oct. 19-20, ASME, AIME, fuels conference, Neil House, Columbus, Ohio.

Nov. 13-18, American Society of Mechanical Engineers, Diamond Jubilee annual meeting, Congress, Hilton, and Blackstone Hotels, Chicago.

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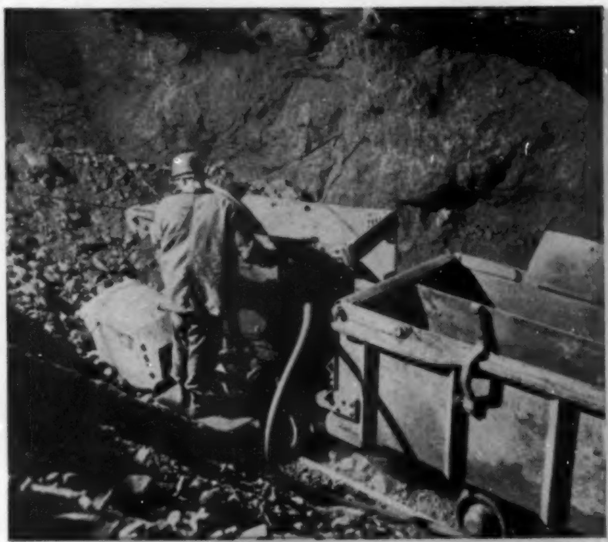

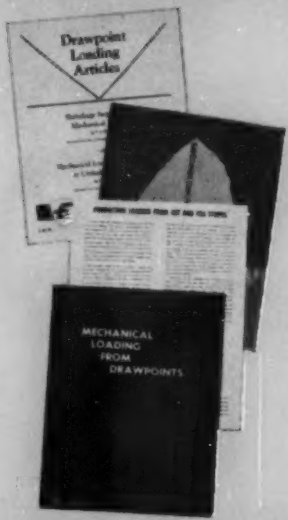

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